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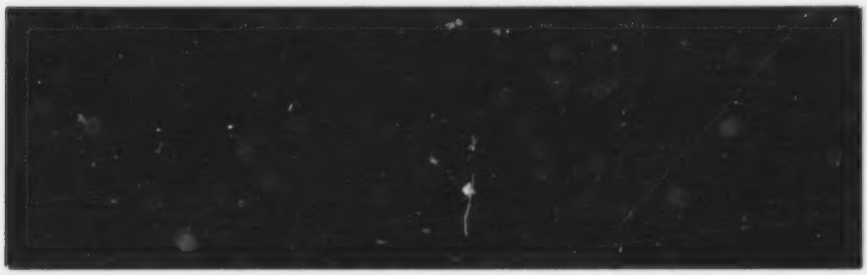

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PRACTICAL AND THEORETICAL ASPECTS OF SULPHIDE DUST EXPLOSIONS

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FOREWORD

As a result of an increasing incidence of sulphide dust explosions in Canadian base metal mines, a concerted explosion control effort by 35 cooperating mines and agencies, has been actively pursued since 1985. These agencies have formed a cooperative technical committee named the "Dust Explosion Control Group." This 'Group' has met twice a year since the inaugural meeting hosted by the GECO Division of Noranda Mines Ltd., held at Manitouwadge, Ontario, in October of 1985. Since this first meeting, there have been several achievements, including:

(1) the production and broad dissemination of CANMET's Special Publication SP87-3, entitled "Sulphide Dust Explosions." This document consolidated the presentations of the 1985 Manitouwadge meeting;

(2) an industry Task Force was formed representing those five mines where a relatively high incidence of sulphide ore dust explosions occur. Moreover, eight mining companies and one explosives company, with CANMET administrative support, contributed to the implementation of a high priority study by the Task Force to carefully document the in-mine effectiveness of current explosion inhibition techniques;

(3) among a number of 'Group' studies, a bench-scale ore-dust explosibility investigation has been completed. It involved cooperation among Noranda Research, CANMET, and the Centre de Recherches Minérales of the Government of Quebec. Original related work had been completed at the University of Sydney, Australia. The explosibility of ore-dusts from a number of mines across Canada, and particularly that of pure sulphide dusts, was determined. These results provide an additional indicator of relative dust explosibility, thereby improving the ability to forecast the likelihood of trouble when developing new ore bodies; and

(4) a special session of the Annual General Meeting of the Canadian Institute of Mining and Metallurgy (CIM) in Edmonton (May, 1988) was devoted to sulphide dust explosion phenomena. On this occasion, eight papers were presented by members of the 'Group'. The Proceedings of the session will be published by the CIM in a special volume.

Finally, a significant milestone was achieved recently with the completion of a CANMET contract to Dr. Roger Enright of the University of Sydney, Australia. The result is this document, entitled "Practical and Theoretical Aspects of Sulphide Dust Explosions." It is intended that this document should be widely disseminated. Accordingly, it is being published as a CANMET Special Publication. It has been written keeping in mind the every day practical needs of the mine operators, as well as those of the regulatory authorities.

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1. INTRODUCTION

1.1 GENERAL

Over the past two decades there has been a significant increase in the number and severity of sulphide dust explosions in the underground workings of massive sulphide deposits in Australia, Canada, Europe and South Africa. While the most severe explosions have occurred in open stopes, a significant number of serious explosions have been associated with development headings and secondary blasting at drawpoints. Fortunately the number of fatalities associated with these explosions has been minimal due to the practice of evacuating mines before stope blasts. However, extensive damage to plant and equipment has occurred during many of the violent explosions and even with minor explosions the production time lost while a mine is checked for the presence of sulphur dioxide is appreciable.

The mining industry has been somewhat complacent in its attitude to sulphide dust explosions. Miners throughout the world recognise that coal dust is explosive and that adequate precautions must be taken to prevent coal dust explosions. However, few mining engineers appreciate the potential for sulphide dust explosions and the dangers associated with them. This attitude can be understood when it is recognised that only three fatalities associated with sulphide dust explosions have occurred in Canadian mines.

Many mining engineers are somewhat surprised to learn that dust from selected portions of a sulphide orebody may be the cause of a dust explosion. Even in mines where minor dust explosions, say as evidenced by lengths of burnt ventilation ducting, have occurred during development work in a new section of orebody the dangers are not appreciated. Indeed, a common attitude is to minimise the dangers and describe the explosions as "sulphur fires", "sulphur burns",

"incidents", or even "second blasts". Rarely are small explosions reported to senior management or attention drawn to the mine planning staff so that at a later date the blasting engineer can be warned of the potential for a sulphide dust explosion. An even more rare occurrence is when the explosion is reported to the appropriate authorities as an unplanned explosion.

1.2 DUST EXPLOSIONS

A dust explosion is essentially a very rapid combustion of a suspension of fine particles, during which heat is generated at a higher rate than is dissipated to the surroundings. This phenomenon is characterised by the sudden development of pressure and a flame front which frequently causes loss of life and extensive destruction of plant and equipment. The explosion leaves in its wake an atmosphere with reduced oxygen content and highly toxic gases. A dust explosion occurs when the following four conditions are satisfied simultaneously:

1. A combustible solid in a finely divided state is dispersed in an oxidizing medium, usually air.
2. The concentration of the dispersed dust is within the explosible range.
3. An external source of ignition with sufficient energy to ignite the cloud is present.
4. A chemical reaction occurs in a confined space.

The development and severity of a dust explosion are functions of the chemical composition, particle size distribution, surface area, density and concentration of the dust. Additional variables are the spatial characteristics of the enclosing space, degree of turbulence of the dust cloud, and the intensity and duration of the igniting

source. The chemical composition of the dust particles is important as an increase in combustible matter will increase explosibility whereas the addition of inert dust particles or water will decrease the dust explosibility.

Under normal working conditions the appropriate concentration o airborne dust is below the minimum explosive concentration; however, dust may accumulate on surfaces in a mine. Two factors must act simultaneously to initiate a dust explosion: a factor which will cause the raising of a dangerous cloud of dust and the appearance of an ignition source capable of igniting the cloud. In a coal mine the usual sequence in the development of a dust explosion is that a local ignition of a methane-air mixture causes a gas explosion and the pressure wave from the explosion disturbs dust from the roof, ribs and floor of the workings. The flame front associated with the gas explosion ignites the airborne dust and a secondary, and more violent, self-sustaining dust explosion is produced (Figure 1.1). The explosion leaves in its wake carbon monoxide and this highly toxic gas may be distributed with fatal results throughout the mine workings.

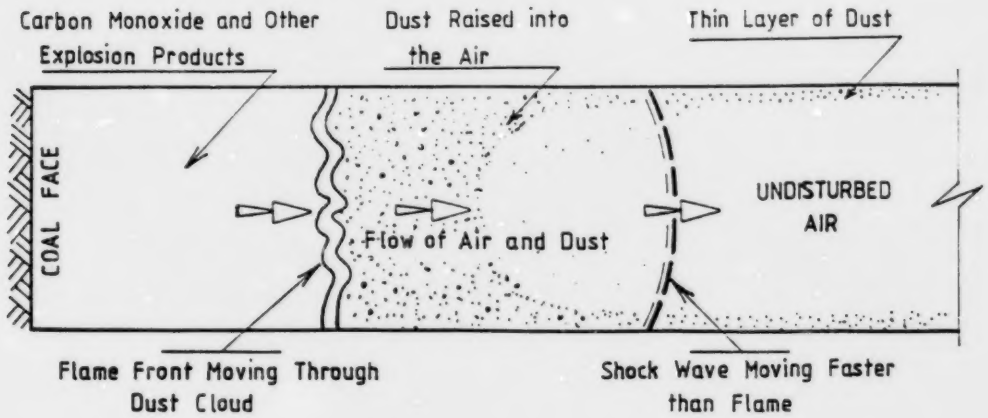

Carbon Monoxide and Other Explosion Products
Dust Raised into the Air
Thin Layer of Dust
COAL FACE
Flow of Air and Dust
UNDISTURBED AIR
Flame Front Moving Through Dust Cloud
Shock Wave Moving Faster than Flame

FIGURE 1.1 - COAL DUST EXPLOSION

1.3 SULPHIDE DUST EXPLOSIONS

In a metalliferous mine the generation and ignition of a dust cloud are usually associated with the use of high explosives in stoping operations, development work or in secondary blasting. A combustible dust cloud may be formed during the breakage and subsequent movement of the rock. In addition the air blast and ground vibrations generated during blasting may disturb dust previously deposited on the mine workings. Detonation of explosives during and after the formation of the dust cloud or the burning of undetonated explosives may provide ignition sources.

In a pyritic dust explosion the products of combustion are usually magnetite, Fe_3O_4, or hematite, Fe_2O_3, and sulphur dioxide. Typical reactions are

$$3FeS_2 + 8O_2 = Fe_3O_4 + 6SO_2 + 2462 \text{ kJ/mol.}$$
$$4FeS_2 + 11O_2 = 2Fe_2O_3 + 8SO_2 + 3458 \text{ kJ/mol.}$$

Sulphur dioxide is a highly toxic gas. A concentration of 400 to 500 parts per million (0.05%) is immediately dangerous to life and 50 to 100 ppm is considered to be the maximum permissible concentration for exposures of 30 to 60 minutes. Carbon monoxide, the major cause of loss of life in coal dust explosions, is considerably less toxic than sulphur dioxide and a concentration of 400 to 500 ppm can be inhaled without appreciable effect for 1 hour.

1.3 POTENTIAL FOR A MAJOR DUST EXPLOSION

There is a very real possibility that a major sulphide dust explosion with a large number of fatalities may occur at a mine even when basic safety precautions are in force at the mine. Consider a mine producing from several levels with stope blasts restricted to the times when the mine is evacuated. Secondary blasting at

drawpoints is permitted during the shift provided miners are in underground meal rooms or at shaft stations. A major disaster could develop as follows:

1. During the shift a drawpoint "hangs up" and it is decided to blast the drawpoint during the meal break.

2. The surfaces of the extraction cross-cut are washed down prior to blasting at the drawpoint.

3. Several sticks of explosive are pushed up the drawpoint, past the washed down rock.

4. Blasting at the drawpoint causes a minor dust explosion. This primary dust explosion causes a secondary dust explosion in the stope feeding the drawpoint.

If it is assumed that the flame front associated with the explosion is confined to the stope, then miners in the meal rooms would be safe from the heat and pressure effects of the explosion. A large volume of sulphur dioxide would be generated during the dust explosion and the hot gaseous mixture may have sufficient pressure to travel through the mine workings and reverse the air flow in certain sections of the mine. Miners in the lunch rooms on several levels could be exposed to high concentrations of sulphur dioxide. Personnel proceeding to the shaft stations after detecting sulphur dioxide could be overcome by the gas while waiting for the cage to hoist them to safety.

The preceding scenario is based on incidents which have occurred at several mines. For example, the toxic gas flow associated with the sulphide dust explosion at the Geco mine travelled around 7400 feet before killing one miner and seriously injuring two other miners. In one section of the mine the ventilation was reversed during the gas flow (Byberg,

1986). Secondary dust explosions in longhole stopes have occurred at two Canadian mines when attempts were made to clear drawpoints with explosives.

The number of fatalities caused by an explosion is usually taken as a measure of the magnitude of the explosion. This method is correct with reference to the terms of the disaster, but it is not exact for the terms of the explosion. However a more appropriate measure of the magnitude of an explosion is the range of the explosion, taking into account travel of the flame front and its velocity, the effect of the blast and the distance the post-explosion toxic gases travel after the explosion. When the range of the explosion is used to define its magnitude then several of the sulphide dust explosions that have occurred in Australian and Canadian mines should be classified as major explosions.

WOODCUT OF AN UNDERGROUND METHANE EXPLOSION

2. COAL AND INDUSTRIAL DUST EXPLOSIONS

2.1 COAL DUST EXPLOSIONS

Coal dust explosions have been a common occurrence in underground coal mining. A review of some explosions and the measures introduced to prevent future explosions may suggest techniques applicable to the prevention of sulphide dust explosions in the hard rock mining industry.

During the eighteenth century it was assumed that methane alone was responsible for the frequent catastrophic explosions that occurred in coal mines. In the worst explosions hundreds and sometimes even thousands of miners were killed in a matter of seconds or minutes. Mining communities or villages were often abandoned following such explosions as the majority of the local men and boys perished in the disasters.

Suspicion that coal dust could play a part in a mine explosion developed during the nineteenth century when coked coal dust was found after explosions. Faraday recognised the danger of coal dust following an explosion in the Haswell Mine in 1844 and stated that coal dust had played an active and substantial part in the explosion. In 1879 a Commission was appointed in the United Kingdom to investigate the problem of explosions in mines. The Commission's report of 1894 stated that the coal dust could explode by itself without the participation of methane. Despite this report, and full scale experimental tests with coal dust, the mining industry failed to recognise the danger of coal dust. This lack of recognition may have been influenced by economic considerations. At the time it was

impossible to mine coal without generating large masses of coal dust and no methods were available to render the dust harmless.

The crucial occurrence which started investigations into coal dust explosions and the ways to suppress them was the disastrous explosion in the Courrier Mine in France on March 10th, 1906 (Cybulski, 1975). The explosion covered an area around 5km long and 1.5km wide. No methane was present in the mine and the only conclusion was that coal dust alone was responsible for the disaster. A major contributing factor to the extent of the disaster was the absence of any means of suppressing a coal dust explosion. A direct conse- quence of this disaster was the establishment of research activities in several countries to investigate the causes and means to prevent coal dust explosions.

Coal mining regulations introduced in the United Kingdom in 1920 required the spreading of incombustible dust, often called stone dust, in the mine workings. The use of stone dust resulted in a decrease in the number and severity of dust explosions as shown in Figure 2.1. In the period 1931 to 1935 a number of serious explosions occurred in the United Kingdom and severe requirements were introduced concerning stone dusting.

Extensive research has continued on the causes and development of coal dust explosions. Preventive measures introduced in the Twentieth Century were directed towards inerting the coal dust with stonedust, eliminating the possible ignition sources by prohibiting naked flames, the introduction of flame proof electrical equipment, restrictions on the type of explosives, limits on the number of shots fired, the intervals between shots, water infusion, and the use of stonedust and water barriers. Despite these precautions the number and severity of dust and gas explosions have remained at unacceptable levels. Table 2.1

shows that in the period 1930 to 1960 the number of explosions, in which 30 or more miners were killed, was 104 with a total of 9443 fatalities (Cybulski, 1975).

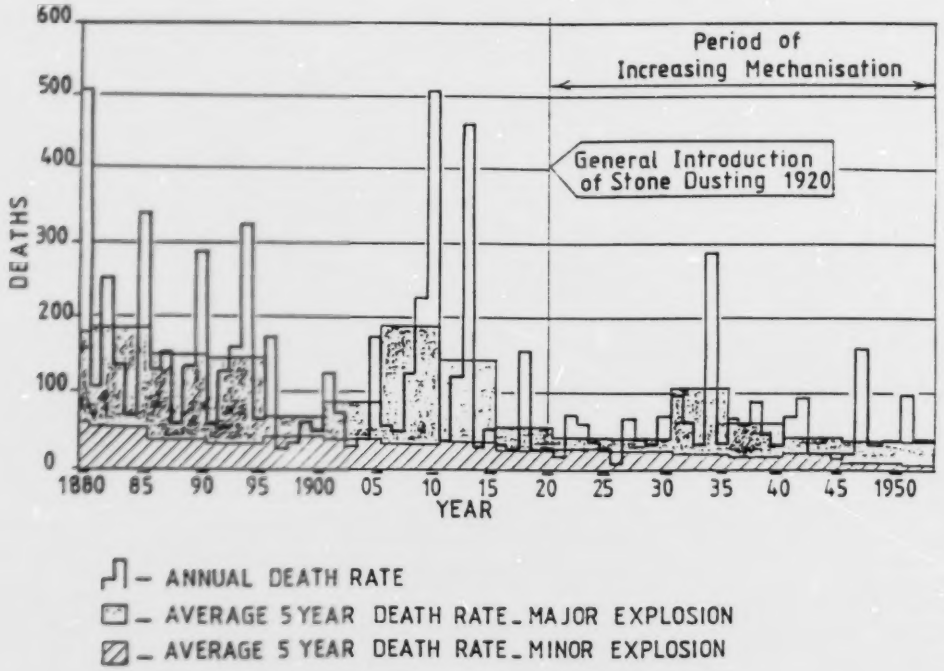


FIGURE 2.1 NUMBER OF PERSONS KILLED IN COAL MINE EXPLOSIONS 1880 - 1950 (Tideswell, 1955)

Since 1960 the number of coal and gas explosions has not been significantly reduced. Cybulski (1975) estimated that more than 3200 miners were killed in explosions between 1960 and 1972. The following examples show that mine explosions are still a major cause of underground fatalities.

Table 2.1: Explosions in Coal Mines with more than 30 Fatalities (Cybulski, 1975)

Country	Number of Explosions	Fatalities
Belgium	3	124
Canada	2	84
France	2	97
Germany	13	1257
India	6	597
Italy	2	228
Manchuria	1	1527
Japan	40	3258
Spain	1	34
Great Britain	11	879
United States	16	996
South Africa	7	362
TOTAL	104	9443

Methane and coal dust explosion in 1962 in the Luisenthal Mine in the Federal Republic of Germany in which 299 miners were killed.

Coal dust explosion in 1963 in the Mamano Mine in Japan with 331 fatalities.

Methane and coal dust explosion in 1965 in the Bhori Mine in India with 306 deaths.

Methane explosion and fire in 1969 in the Farmington Mine in the USA with 78 men killed.

Explosion in 1970 at the Breza Mine, Yugoslavia with 51 deaths.

Coal dust explosion in 1972 in Zimbabwe with 426 deaths.

A brief analysis of three of the disasters listed above will indicate some of the main factors which contributed to the explosions, the large number of fatalities and changes in Government Regulations.

LUISENTHAL COLLIERY

The 1962 explosion at the Luisenthal mine resulted in the deaths of 299 miners. Before the disaster the mine was obliged to use:

* stonedusting to increase the percentage of incombustible content of the dust mixture to 50%.
* install stone dust barriers although the location of the barriers was not defined precisely.

The cause of the explosion was thought to be the ignition of a methane layer, either by cigarettes or a damaged cap lamp, which carried the flame to a methane accumulation. This accumulation exploded and resulted in a secondary coal dust explosion. The great range of the explosion was explained by the fact that it passed through workings containing coal dust and that in the stonedusted areas a reduction in the combustible content of the dust to 50 percent was inadequate to prevent an explosion. It is now generally considered in current mining practice that the incombustible content of a dust should be greater than 75 percent in order to prevent a dust explosion. Other contributing factors to the explosion were the incorrect design and location of the stone dust barriers (Cybulski, 1981).

MIKAWA COLLIERY

A plan of part of the Mikawa Colliery, which produced 187,000 tonnes per month and employed 3923 miners, is shown in Figure 2.2. Mining operations were carried out on the

350m and 450m levels with Inclines 1 and 2 used as intake airways and the Shinko shaft as the return airway. The No. 1 Incline was 2000m in length with a slope of 12°, 6.4m wide by 4.5m high. This incline was partitioned with one section used as a haulage belt conveyor and the other section used for track haulage of men and material. Electrical cables were also installed in the track section.

The explosion occurred between shifts, when 1388 miners from the first shift and 692 miners from the second shift were underground. At the time of the explosion the conveyor in Incline 1 was stopped but eleven mine cars were being lowered in the other section. At 1180m in from the surface a coupling between the third and fourth car was broken and the eight uncoupled cars hurled down the decline where they were derailed at 1576m and ran an additional 54m before coming to a stop. The derailed cars cut electrical cables in the raised dust cloud and electrical arcing initiated the coal dust explosion. The Mikawa mine was a non-gassy mine and coal dust alone provided fuel for the explosion.

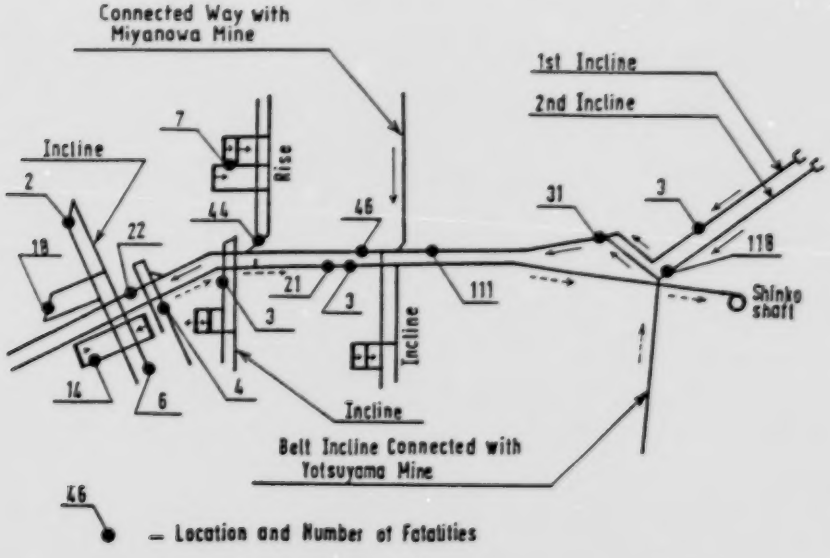


FIGURE 2.2 PLAN OF PART OF MIKAWA COLLIERY
(Cybulski, 1970, after T. Suzuki)

Incline No. 1 was extensively damaged while the other workings were only slightly damaged. In the explosion 458 miners were killed, 25 seriously injured and 817 slightly injured. Only 20 miners were the direct victims of the blast of the explosion; the other 438 miners, many at remote locations from the blast, were fatally poisoned by carbon monoxide (Cybulski, 1981).

FARMINGTON MINE

The 78 fatalities that occurred during the Farmington Mine methane explosion and fire in 1969 so disturbed the public that a Senate Committee was constituted to investigate the causes of the disaster. As a result of the findings of the Committee the mining legislation and system of enforcement in the United States were radically changed.

2.2 INDUSTRIAL DUST EXPLOSIONS

The potential for a dust explosion exists in virtually all industrial processes where combustible particulate matter is either manufactured or handled. Typical dangerous materials include agricultural products, foodstuffs, chemicals, pharmaceuticals, wood products, plastics, dyestuffs, metal alloys and non-metals.

The first well-documented grain dust explosion occurred in an Italian flour mill in 1785. At that time it was thought that the dispersed dust had produced an inflammable gas which had subsequently ignited. With increased production and mechanisation in the grain handling and manufacturing industries the number of dust explosions increased. Statistics related to dust explosions are limited but figures presented in Table 2.2 for the period 1900-1956 in the U.S.A. and Canada (National Fire Protection Association, 1957) indicate the relative frequency of materials involved in dust explosions.

Table 2.2: Frequency of Materials Involved in Dust Explosions in the U.S.A. and Canada (1900-1956)

Material	U.S.A. (%)	Canada (%)
Agricultural	54	56
Wood products	26	15
Metal	7	8
Plastics	5	2
Coal (excluding mining)	5	11
Others	3	8

The number and severity of industrial dust explosions that have occurred are staggering. In the U.S.A. up to December 1956, 1120 dust explosions had occurred, 675 people had been killed, 1770 had been injured and the estimated property damage was in excess of $100 million (Anon. 1978). Bartknecht (1978) using data obtained from insurance companies found that more than 4000 dust explosions were reported in Europe between 1960 and 1972.

In the United States 220 grain dust explosions occurred between 1958 and 1977, which resulted in 48 deaths and 500 injuries (Hall, 1978). Two of the most serious dust explosions occurred in Westwego, Louisiana, and Galverston, Texas. A total of 54 people were killed and the property damage exceeded $60 million (Lahrop, 1978).

The explosion at Westwego killed 35 persons, destroyed the head house, office buildings and 36 concrete silos 35m in height and 8m in diameter. The violence of the explosion at Galverston was shown by the destruction of a reinforced concrete conveyor tunnel 450mm thick and located 1m below the ground. A large ditch was created by the destruction of this tunnel and this cut the only road giving access to the site for firefighting.

These two dust explosions received international coverage and the devastating consequences of exploding dust were immediately recognised by many in the industry, who had chosen previously not to acknowledge the problem, or were totally unaware of it.

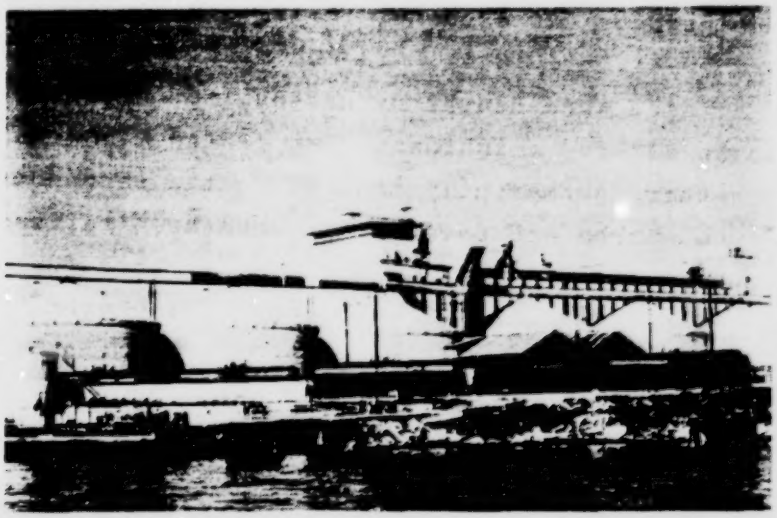


PHOTO 2.1 CONCRETE BUILDINGS, CONCRETE SILOS AND STEEL STORAGE BINS BEFORE GRAIN DUST EXPLOSION

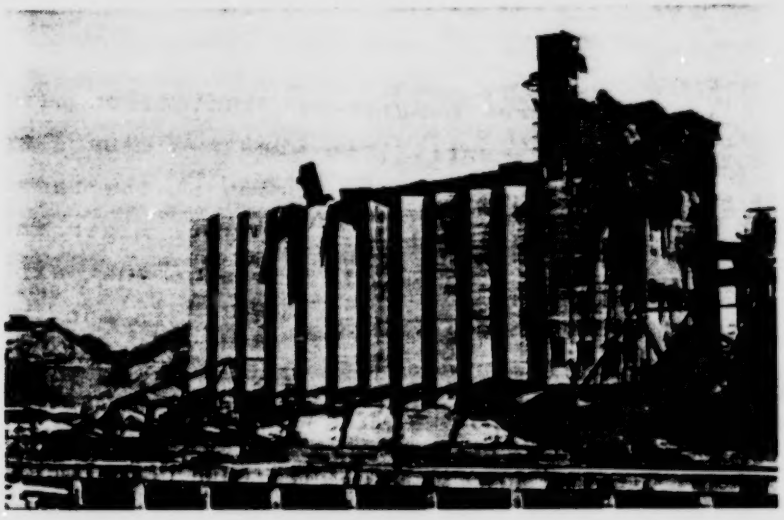


PHOTO 2.2 REMAINS OF BUILDING SHOWN IN CENTRE OF PHOTO 2.1. SILOS TO LEFT OF BUILDING DEMOLISHED, STEEL BINS RUPTURED

3. SULPHIDE DUST EXPLOSIONS

3.1 INTRODUCTION

Some of the earliest reported fatalities due to sulphide dust explosions occurred in 1924 in the United States (Gardner et. al. 1926). In one mine, three men working in a raise were caught by gasses from blasting a drift round in massive sulphides. Two of the men were killed and a third permanently injured. At a second mine one fatality occurred and several men were hospitalised due to the sulphur dioxide generated during a pyritic dust explosion. At a third mine about 35 men were caught by gases following a blast in heavy sulphides. Of the 35 men, seven were hospitalised; one became a fatality, and three were long time disabilities.

The first reported case of a fatality due to the flame front associated with a pyritic dust explosion occurred in 1926 in the United States (Gardner and Stein, 1928). Three men died as a result of burns and a fourth was seriously injured.

The available literature gives no indication of further sulphide dust explosions until 1969 when two miners were killed following a massive dust explosion at the Langsele mine in Sweden (Enright, 1986). In 1985 one miner was killed and two seriously injured following a dust explosion at the Geco mine in Canada (Byberg, 1986). In one incident, not detailed in the literature, two miners were killed in the 1960's following sulphur dioxide inhalation. The dust explosion occurred when a drift round was fired and destroyed timber sets for half a mile along the drive.

3.2 REVIEW OF SOME SULPHIDE DUST EXPLOSIONS INVOLVING FATALITIES

Deaths associated with sulphide dust explosions may be caused by injuries due to shock waves, burns from the flame front and the inhalation of poisonous gases formed during the explosion. Fatal injuries due to the shock wave and flame front occur within seconds of development of the dust explosion whereas death due to gas poisoning may occur some time after the dust explosion. A summary of the reported fatalities and the probable cause of death is given in Table 3.2.

Table 3.2: Reported Deaths and Serious Injuries due to Sulphide Dust Explosion
(Probable Cause given in brackets)

Date	Country	Fatalities		Injuries	
1924	U.S.A.	1	(SO_2)	6	(SO_2)
1924	U.S.A.	1	(SO_2)	1	(SO_2)
1924	U.S.A.	2	(SO_2)	1	(SO_2)
1926	U.S.A.	3	(burns)	1	burns
1960's	Canada	2	(SO_2)		-
1969	Sweden	2	(SO_2)	2	(SO_2)
1985	Canada	1	(SO_2)	2	(SO_2)
Total		12		13	

U.S.A. MINES

During 1924 a number of lives were lost and other men incapacitated by the toxic effects of gases produced in blasting massive sulphides. The following details are taken from the U.S. Bureau of Mines Report of Investigations by Gardner et. al. (1926).

In one mine, three men working in a raise were caught by gases from blasting a drift round in massive sulphide on the level. The round in the drift was shot ahead of time and it was necessary for the men to travel through the gases to the intake air course. Two of the men lost their lives, and the third was injured, apparently for life. Other men at the same mine have been caught in the gases at blasting time and seriously injured.

At a second mine it was necessary to send a number of men caught in the gases from blasting in sulphide ore to hospital. One man who was seriously gassed had not recovered in a year's time. Another man similarly gassed soon died of influenza.

At a third mine about 35 men were caught by the gases from blasting a round in heavy sulphides, which was shot ahead of time. Of the 35 men, seven developed hospital cases; one of these was fatal, and three were long time disabilities.

Gardner and Stein (1928) investigated a sulphide dust explosion that occurred in 1926 in the United States in which three men were killed and a fourth severely burnt by the explosion. The accident was caused by a blast in a surface churn drill hole designed to break through to raise from the underground workings of a pyritic orebody assaying around 40 percent sulphur. A sketch of the underground workings is shown in Figure 3.1. The previously sprung churn drill hole was loaded with 68kg of 35 percent strength dynamite and prior to firing men were withdrawn from the 300, 400, 600, 700, 900 and 1000 levels. Due to an oversight, men working on the 500 level were not notified of the blast. On the 800 level two men were working 15m along

a cross-cut 23m from No. 7 raise and a third man was working 15m up a raise around 15m past No. 7 raise. The men on 800 level were not considered to be in any danger.

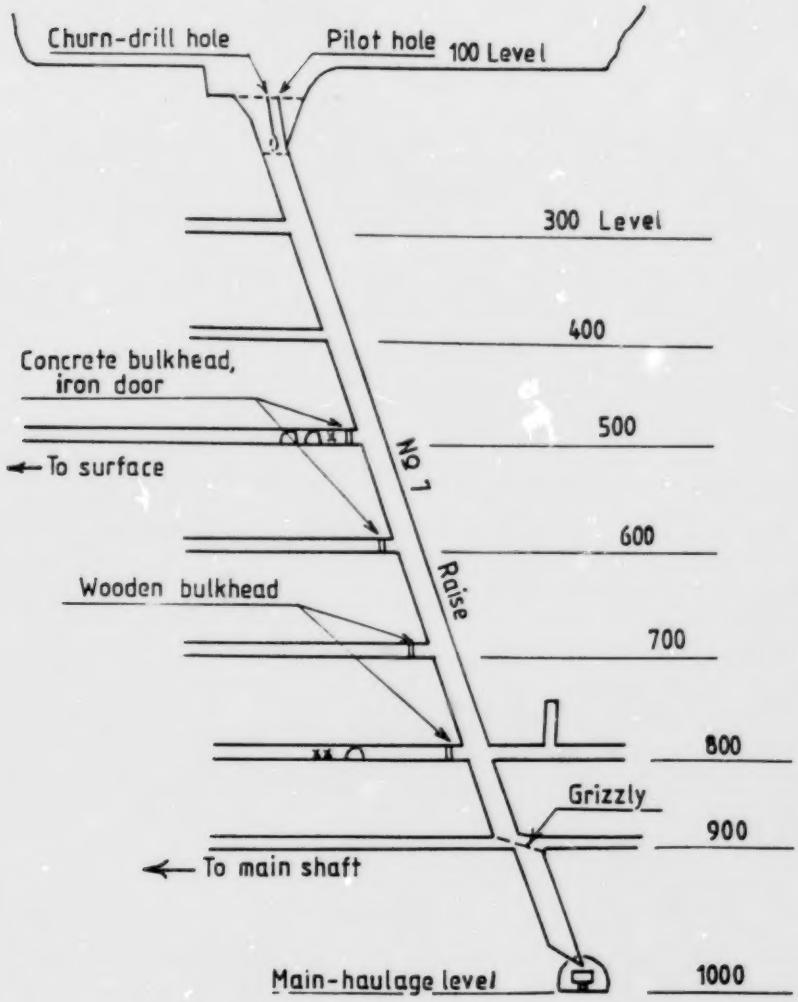


FIGURE 3.1 DIAGRAMMATIC SKETCH OF NO. 7 RAISE
(After Gardner & Stein, 1928)

When the churn drill hole was fired a second explosion was heard a short time afterwards. A supervisor on the 500 level reported a stream of fire poured out of the raise and one man working 23m from the raise was killed. On the 800 level the two men working in the cross-cut suffered severe burns and died the following day. The man working in the raise suffered serious burns when he made his way to the main haulage drive. It is apparent from Gardner and Stein's report that a strong pyritic dust explosion occurred in the mine. The flame front travelled 76m down the shaft, ruptured a wooden bulkhead and proceeded 29m horizontally to kill two men on the 800 level. The settled sulphide dust from the explosion was iron red and a sample collected several hundred metres from the raise on the 800 level assayed 19 percent sulphur compared with 45 percent sulphur in the orebody.

LANGSELE MINE (Sweden)

A severe sulphide dust explosion occurred at the Langsele Mine, Sweden, on 23rd May, 1969. Two miners were killed and extensive damage caused on 160, 210, and 260m levels (Rova and Lovgren, 1969). Production commenced at the mine in 1956 and at the time of the accident a total of 450,000 tonnes per year were produced using sub-level stoping. The orebody is a complex pyrite ore with the following average analysis: 0.85% Cu, 0.10% Pb, 3.84% Zn and 38.7% S.

A sub-level stope with holes loaded from the 176, 210, 232 and 250 sub-levels was fired on the 13th May, 1969. The 45mm diameter holes were loaded with ANFO (prills) and dynamite primers. All holes on the 210 and 232 sub-levels and one third of the holes on the 176 sub-level misfired. An attempt was made to fire the remaining holes on the 14th but only half of the holes detonated. The remaining holes

containing 800kg of ANFO were checked and prepared for firing at the end of the afternoon shift on the 23rd of May. The main blast was to be initiated on the 210 sub-level shaft station and the two shot-firers were to travel to the 260 level to fire a raise. The cage was positioned at the 210 sub-level and the mine was evacuated except for the two shot-firers. A section through the mine showing the levels and the stope outline are shown in Figures 3.2 and 3.3.

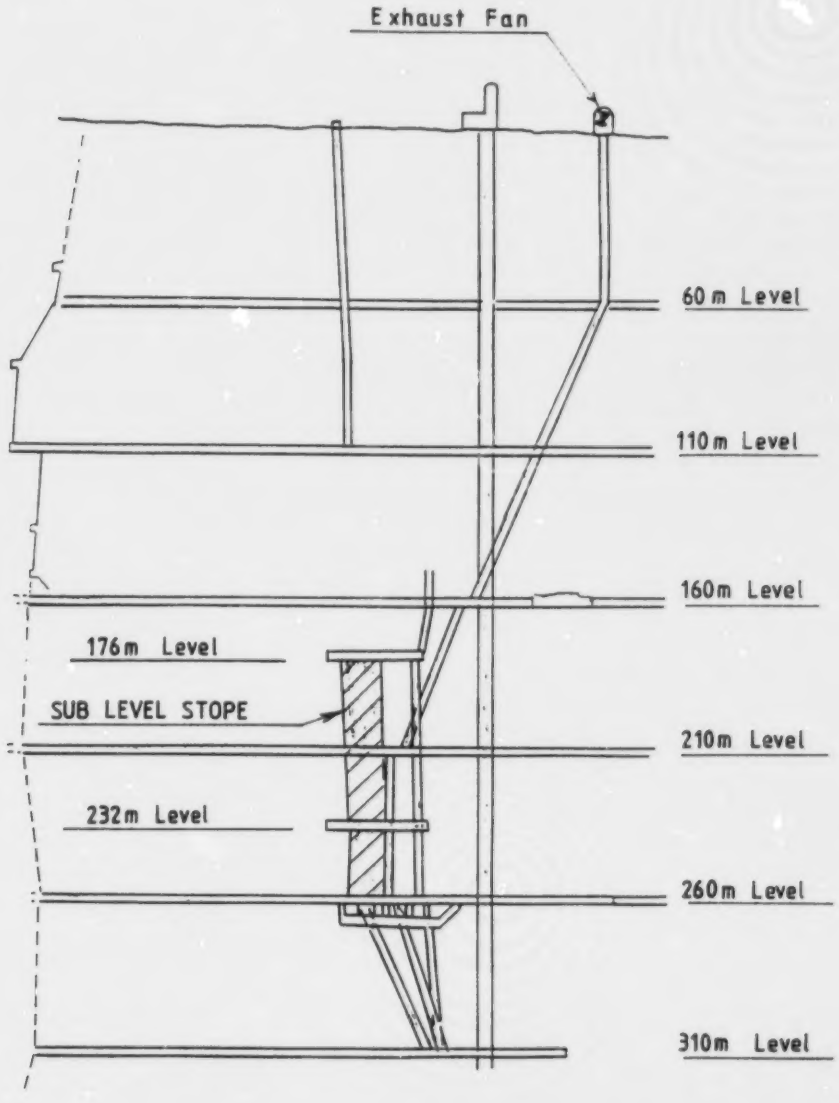


FIGURE 3.2: SIMPLIFIED SECTION THROUGH LANGSELE MINE

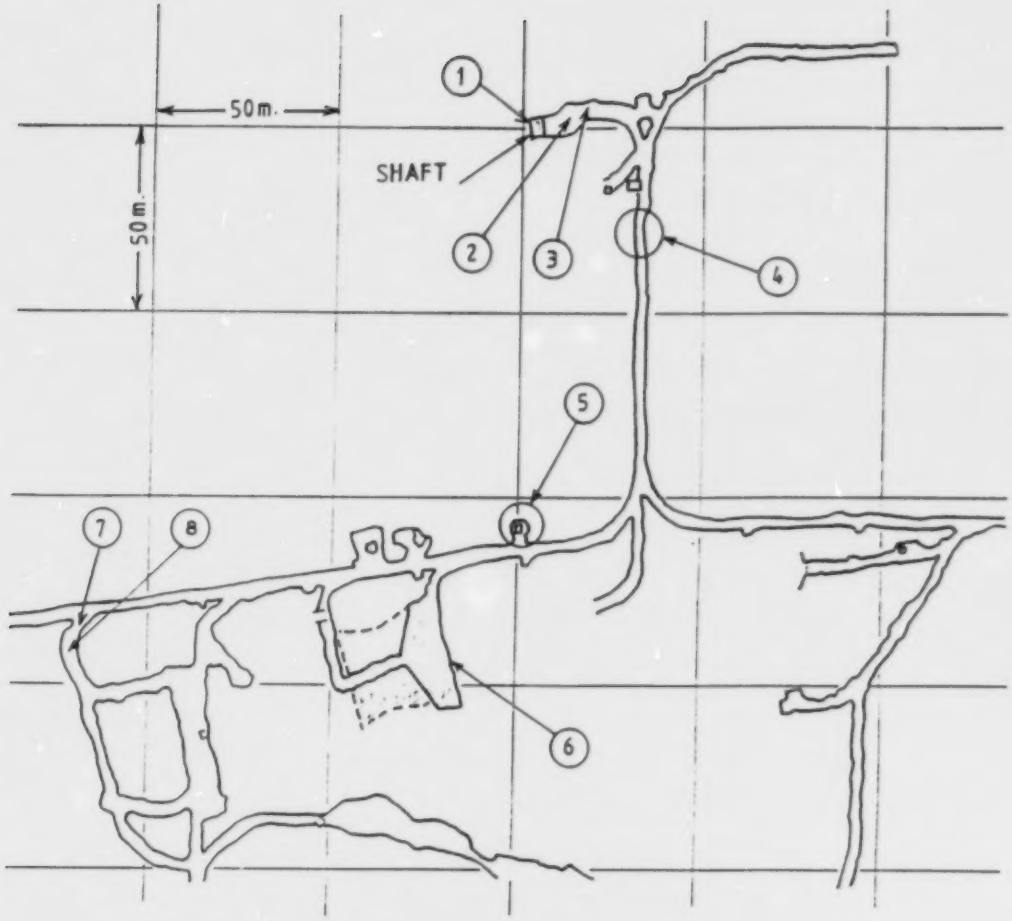


FIGURE 3.3 SIMPLIFIED PLAN 210 LEVEL LANGSELE MINE

The following account is from Foreman Bergkirst's report. "I listened to the sub-level benching blast (from near the head-frame) and determined that the blast had gone

well. Soon after there was heard a hurricane-like sound, then sulphur smelling smoke, small stones and dust blown through the shaft. Several men were required to open the hoist room door and help the driver out. The intense blowing throughout the shaft continued for about 4 minutes."

Bergkirst put on a pneumatic mask, ordered the hoist driver to do the same in order to stay in the hoist room and proceeded via the ladder to the 210m sub-level There he found the two miners lying lifeless in the cage. The cage was raised to the surface where the men were declared dead.

The shot firers had intended to take the exploder from the 210m sub-level to the 260m level to fire a raise. The exploder was found 10m from the shaft (Photo 3.1) with the firing cables still connected to it. This indicated that the miners were someway forewarned of impending danger and proceeded without delay to the shaft. They reached the cage but apparently were so rapidly overcome by sulphur dioxide that they were unable to signal the winder driver to hoist them to safety at the surface. The location of the shaft and exploder on the 210m sub-level is shown in Figure 3.3.

Extensive blast damage and evidence of the flame front was found on the 210m, 260m, and to a lesser extent, on the 160m levels. The range of the explosion on the 210m sub-level is shown in Figure 3.3 and the violence of the explosion is shown by Photos 3.2 and 3.3 taken on the 210m sub-level and Photo 3.4 taken on the 260m level.

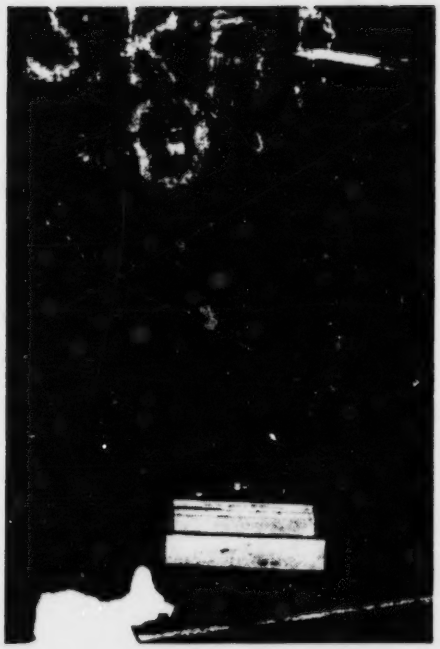


PHOTO 3.1 Exploder with firing cables still connected (210m level)

PHOTO 3.2 Sheet metal damaged doors about 12m from the shaft (210m level)

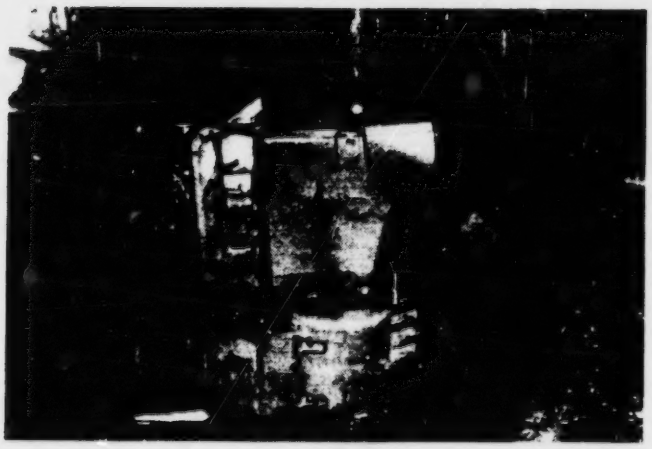


PHOTO 3.3 5 tonne diesel loco and granby truck derailed and forced up drill drive.

PHOTO 3.4 Damaged service equipment on 260m level.

GECO MINE (Canada)

Fifty million tons of a complex sulphide ore were mined from the Geco orebody before the first sulphide dust explosion occurred. On a few previous occasions sulphur dioxide had been detected after blasting but the presence of the gas was attributed to the use of explosives. The dust explosion occurred on October 8, 1985, when a slot blast on the 2450 level was fired. The mine had been evacuated prior to the blast except for three men located on the 2650 level approximately 7400 feet (2256m) from the stope. One miner was killed and two seriously injured by sulphur dioxide produced during the dust explosion. The slot blast was in the 2646 Pillar and a section through the mine showing the location of the pillar and the three men is shown in Figure 3.5. The following description of the events prior to the dust explosion and the damage caused by the explosion are from a paper presented by Byberg (1986).

"The 2646 Pillar contained block contents of 40.8% pyrite, 3.5% chalcopyrite, 5.7% pyrrhotite and 9.8% sphalerite. The total sulphur content was estimated at 28.3%. At the mine, slots are created the full height of the mining blocks so as to produce an optimum void of 20% of the block volume. Development for mining the 2646 Pillar began in 1983 and by October 8, 1985, was completed, as was 80% of the slot blasting. A side view of the 2646 Pillar and the void is shown in Figure 3.4.

On October 8, 1985, at 11.55 p.m. the 23rd longhole void blast, with 4.5 inch (114mm) downholes and 3 inch (76mm) up holes, was fired from the surface. As well, but not a common occurrence, the 2346.5 drop raise immediately above the void blast was fired sequentially. The longholes were charged with 4510 lbs. (2046 kg) ANFO and 330 lbs. (150

kg) water gel which were initiated with NONEL NO. 4 to 19 detonators. The drop raise was charged with 110 lbs. (50 kg) water gel and L.D. Electric No. 1 to 4 detonators.

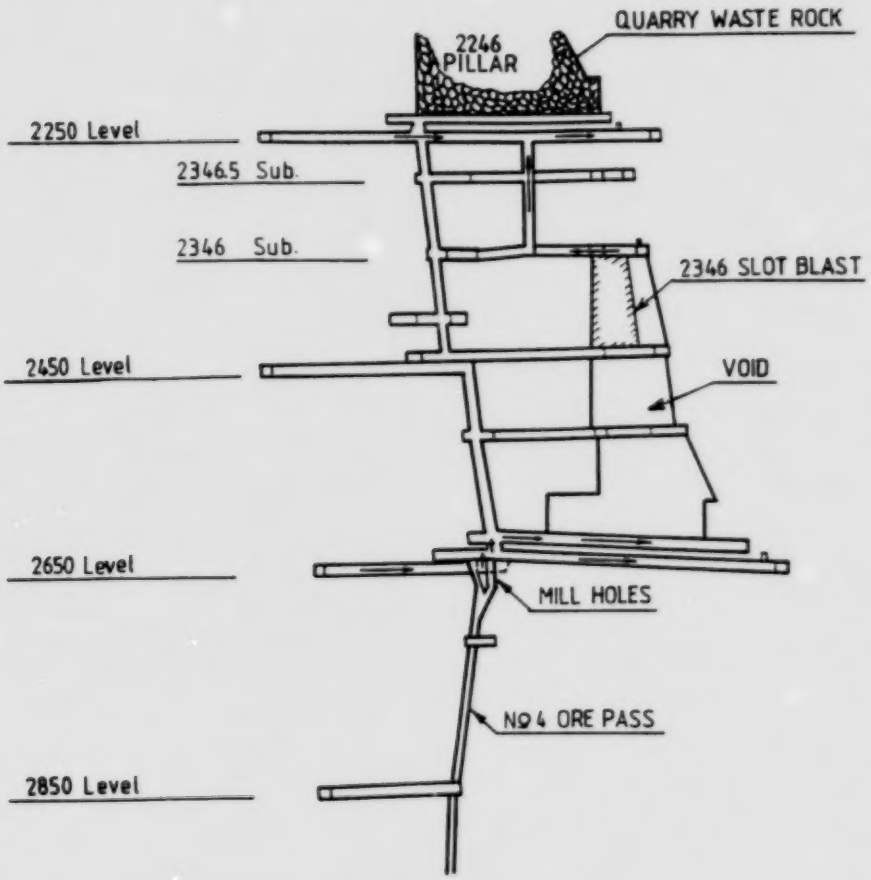


FIGURE 3.4 2646 PILLAR - SIDE VIEW SHOWING VOID AND 2346 BLAST.

The blast was fired from the surface between the 4-12 and 12-8 shift change when the 4-12 crews had been hoisted and prior to the 12-8 shift being lowered underground. The detonation of the slot blast and the subsequent detonation of the drop raise triggered a major sulphide dust explosion. The explosion, as evidenced afterwards, occurred on the 2346 sub-level and the flame front may have in fact reached the 2650 level and ore pass and burned within the cavity also. Extreme heat was evidenced on the 2346 sub-level and to a

lesser degree on the levels below. Plastic borehole plugs, lead wire, telephone and combustible materials were either badly burned or deformed by heat. Spontaneous combustion ignited a fire in the 2346 sub-level which burned until self extinguished. So intense was the heat in this area that the back and walls contained a white residue.

The explosion itself was of such magnitude that the ventilation was reversed momentarily. On the 2650 level the incasting 40,000 c.f.m. (189m^3/s) had no effect on the explosion. A 1.5 inch (38mm) ventilation door located in the 2646 Cross-Cut 400 ft. (122m) from the blast and open at time, was ripped from its concrete moorings and deposited in rubble against the main drift wall 50 ft. (15m) away. Large pieces of it were found up to 540 ft. away against the 40,000 c.f.m. travelling in the main drift. Air ripples on the drift floor similar to waves on a sand beach were found throughout the 2250, 2450 and 2650 levels, however, they were more prevalent on the 2650 level, where the windswept evidence was easily detected up to 1400 ft. (427m) from the blast also towards the 40,000 c.f.m. ventilation flow. The manway between the 2250 level and 2346 sub-level also was damaged and required repairs. A heavy 'sootlike' residue was found throughout the 2246 area from the 2650 and 2250 levels.

The dust explosion occurred as a result of the 2346 void blast. Evidence showed that the flame front originated on the 2646 sub-level and moved throughout the area even to near the 2646 mill holes leading to the ore pass. Whether sufficient sulphide fuel was available to feed the flame front in the ore pass was not known.

The explosion reversed momentarily the ventilation and forced SO_2 gas 1000 ft. (305m) down the ore pass to the No. 4 Crusher. The SO_2 gases reached the Crusher's return air

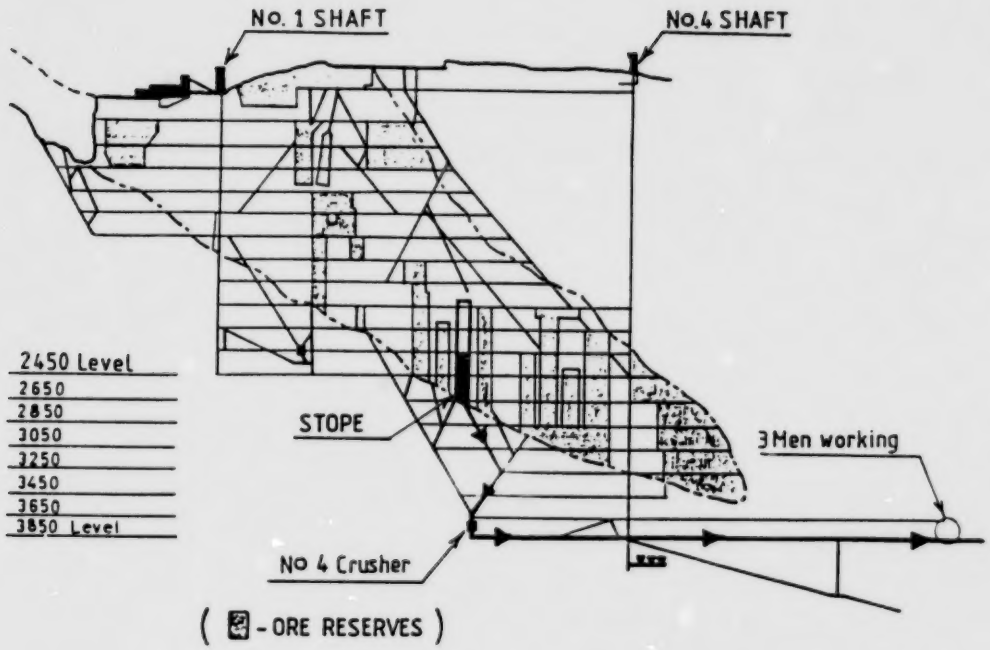


FIGURE 3.5 LONGITUDINAL PROJECTION OF THE GECO MAIN ORE BODY. (After Byberg, 1986)

system and due to the additional volume of air forced from the ore pass, moved rapidly along the 3850 level to the No. 4 Shaft return air column. The quantity of contaminated air far exceeded the normal 17,000 c.f.m. (8M³/s) flowing from the crusher (7000 in line and 10,000 free flow). Consequently some of the contaminated SO_2 bearing air was pulled into the 44,000 c.f.m. (20.8m³/s) fresh air fan at the 4 Shaft Station and forced east to the development heading where the three men were working. One miner was fatally injured. The remaining two who sustained respiratory injuries provided themselves with air from a compressed air header within 300 ft. (91m) of the face until mine rescue crews arrived.

First indications of the problem came from the oncoming 12-8 crews in the 3650 Shaft Station and Loading Pocket areas. Concern at the time was understandable since the No. 4 Shaft downcasts 300,000 c.f.m., no blasting had occurred anywhere near the 3650 level, and yet there was blasting gas reported at the shaft. With the injection of stench gas (ethyl mercaptan), the mine was evacuated with the exception of the three miners on the 3850 level. Mine rescue teams were assembled and sent to recover the three workers.

3.3 DUST EXPLOSIONS ASSOCIATED WITH STOPING

ELURA MINE

In November, 1983, a sulphide dust explosion occurred at the Elura mine in N.S.W. Annual production from the comparatively new mine was around 1 million tonne/year and the explosion caused extensive damage to equipment. No personnel were injured as the mine had been evacuated prior to stope blasting. (Enright, 1984).

The dust explosion followed the firing of a single 200mm diameter by 70.8m length blast hole in an open stope in a semi-massive sulphide orebody containing around 31 percent sulphur. The hole had been loaded with four deck charges with delays between each deck. The bottom charge was 339 kg of Tovex extra and a total of 558 kg of ANFO was distributed in the upper three decks. A 200mm length of agricultural limestone was placed above and below each charge and 2.1m of crushed slag stemming between each deck. The top stemming consisted of 200mm limestone, 3.6m of sand and a final 200mm of limestone.

An inspection of the mine indicated that the dust explosion had been contained within the stope while the shock wave had travelled extensively through the drawpoints

and haulage level (Figure 3.6). A Sullair 300 litre/s electric air compressor on the shaft platform, approximately 140m from the nearest drawpoint, was forced 9m towards the shaft and overturned. In the haulage cross-cut next to a drawpoint a Horwood Bagshaw 12 tonne electric load haul dump machine was forced around 20m down the cross-cut before being driven into the wall with severe damage to the vehicle chassis. At the opposite end of the cross-cut a small portable Sullair compressor 25m from the nearest drawpoint was forced around 20m into the wall and seriously damaged. There was no evidence of flame, heat or red dust in the cross-cut drawpoints or main haulage area.

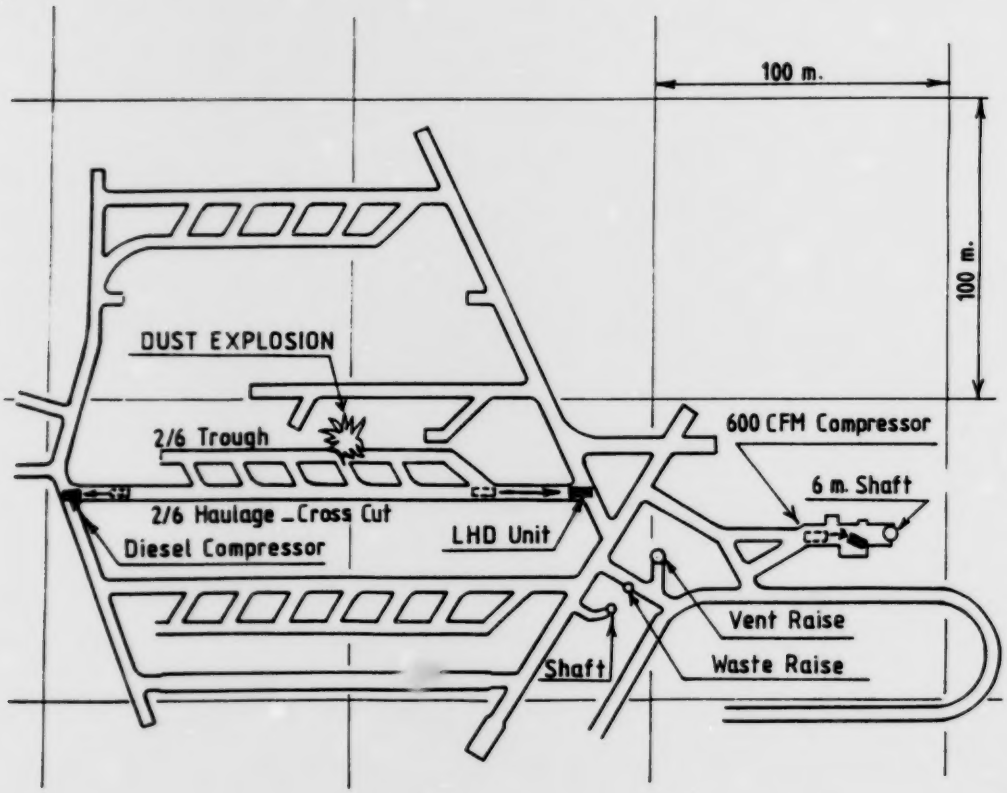


FIGURE 3.6 NO. 2 HAULAGE LEVEL ELURA MINE

RUTTAN MINE (Canada)

In July 1986, a violent dust explosion occurred in the 26J stope at the Ruttan Mine. The 6000 tonnes per day mine had been an underground operation since 1979 and the orebody consists of numerous en echelon massive and semi-massive sulphide lenses. Minerals within the ore lenses included pyrite, pyrrhotite, chalcopyrite, sphalerite and galena. The 26J stope was located in the east lens where previous sulphide dust explosions had occurred.

Blasthole open stoping was followed by delayed backfill using both dry rockfill and hydraulically placed tailings. A raiseborer was used to cut a 5 ft. (1.5m) diameter slot raise and 4.5 inch (114mm) diameter blast holes for the undercut and overcut. The slot was then blasted into the void and finally the remainder of the stope was mined using a sequential vertical slicing technique. The stope was an open system, always having an opening to the top of the stope.

The following description of the sulphide dust explosion in the 26J stope is taken from the paper presented by Nel (1986). "On July 22, 1986 at midnight, the second last stope blast was initiated with some dramatic after effects. The blastholes were loaded with a wooden plug in the bottom, 3 ft. (0.9m) of limestone, nilite charge column and 10 ft. (3.05m) of limestone for the collar. A 10-15 lb (4.5-6.8 kg) bag of limestone was also suspended in the stope. The first indication of a problem became apparent when the man cage returned to the surface after its post blast trial run with a 50 ft. (15m) length of 42 inch(1.07m) diameter flexible tubing. The vent tubing had, prior to the blast, been delivered to the 730 shaft station. During the subsequent shaft inspection, two more lengths of vent tubing were found in the shaft just below 490 metre level, some 240

metres above the 730 shaft station. In addition to this, a regulator in the 370 East exhaust complex had also been damaged. Two 36 inch (914mm) exhaust fans in parallel on the 660 metre were blasted from their mountings and hurled approximately twenty metres down a cross-cut. The fans were located approximately 140 metres above the stope.

Damage to the mine ventilation resulted in a production delay of five shifts for the Lower Mine. Numerous other similar incidents have caused equipment damage, personal injury in the forms of sulphur dioxide inhalation, and production delays."

KIDD CREEK MINES LIMITED

The mining method employed at Kidd Creek No. 2 mine in Timmins, Ontario is an open stope blasthole, in which no pillars are left. Slots are initially opened by boring a 1.2m diameter raise and either 114mm or 105mm holes are then drilled to blast into the raise. The following description of the events associated with the sulphide dust explosion at the mine is taken from D. Duke's report (1987). "On April 1, 1987 at 11.50 p.m. the third blast in a sequence to open up a slot in the 30-571 stope from 30-2 to 30-1 sub-levels was fired from the surface. A total of eight holes were loaded with 425 kg Aquamex II and capped with NONEL short period caps varying between Nos. 1 and 13. The toes of two holes had approximately 12.0 kg of calcium carbonate loaded into each. All holes were stemmed with sand in the collars and a total of 132.0 kg of calcium carbonate was loaded for the entire blast.

As a result of a dust explosion three men suffered gas inhalation and 4 hours production time was lost for the entire underground operation. On the 30-557 cross-cut all the ventilation ducting was either blown down or destroyed. Approximately 8.0m of flexible ventilation ducting was

totally melted, while another 8.0m of spiral ducting only had the steel rings left. NONEL caps stored in a cap magazine located at the entrance to the 30-571 stope had either their shock tubes totally melted off or were badly deformed due to heat. Plastic fences and safety signs in the area were deformed from the heat and a safety vest was melted.

The standard practice at the mine for longhole blasting is to evacuate the mine and a blast is not initiated until all men are on the surface and accounted for. The safety procedure following a longhole blast is to have the mine rescue crews check the mine for smoke or gas before the next shifts proceed to their workplace. After the slot blast at 11.50 p.m. the mine rescue team travelled to the 2600 level via No. I shaft. The level looked normal with no sign of gas or smoke. They then proceeded to the 2800 level where they found the lunchroom normal and notified their supervisor. The men at the surface were then released to travel to the main lunchrooms only. The rescue team continued to check the mine and at 12.32 a.m. detected sulphur dioxide coming from a cross-cut and going into a crusher station. They notified the supervisor and were instructed to get all men out from underground.

Meanwhile, around 12.27 a.m., 10 men were lowered in the No. 2 shaft to the 2600 level. While heading out to the lunchroom they encountered smoke and a whiff of sulphur dioxide. They returned to the shaft area, notified their supervisor, travelled to the 2800 level where they were told to return to the surface immediately.

At approximately the same time, 12.27 a.m., as No. 2 cage was letting off men on the 2600, 4 men were travelling in the No. 1 shaft. Two men left the cage at the 2600 level, encountered smoke and phoned the other 2 men still travelling in the cage. These men then proceeded to the

conveyor area where they encountered fumes with a very strong smell of sulphur. All three men then evacuated the conveyor area. Two of the men were treated for gas inhalation and then sent to hospital. Following the clearing of the mine by mine rescue personnel the men returned underground at approximately 4.00 p.m."

The dust explosion at the Kidd Creek mine warrants consideration as:

* 132 kg of calcium carbonate were placed in 8 blast-holes 104mm in diameter and around 12m in length. The holes were charged with a total of 425 kg of Aquamex II.
* Three men on the 2600 level suffered from sulphur dioxide inhalation due to a dust explosion on the 3000 level. The men encountered the sulphur dioxide 40 minutes after the dust explosion. It is obvious that if safety precautions had not been in effect at the mine then many more personnel may have been affected by gas inhalation.

FOX MINE (Canada)

The Fox Mine has been producing copper/zinc sulphide ore since 1970 at a rate of approximately one million tons per year and prior to 1978 all production was derived from longhole open stopes. A trial stope using bighole mining methods was started in June 1978 so as to eliminate two sub-levels between 400 level and the surface. When production blasting was started in the first stope, the biggest problem encountered was the generation of large volumes of sulphur dioxide as a result of secondary sulphide dust explosions. This gas resulted in lost production time and the explosions caused minor damage to underground installations and to the headframe. (Du Russel, 1981).

The 4-5 stope was mined from the 400 ft. drawpoint level using 6.5 inch (165mm) diameter blast holes. A 10 ft. (3.05m) square slot raise was brought up using vertical crater retreat (VCR), and the rest of the stope was slashed. On January 29, 1979 a secondary dust explosion occurred in the stope. Prior to stope blasting twenty-four, 6.5 inch (165mm) diameter holes had been loaded with 165 pounds (75 kg) of ANFO and 14 pounds (6.4 kg) of Hydromex M210U. A wooden plug was placed six feet (1.8m) above breakthrough in each hole. Approximately two to three pounds (0.9-1.4 kg) of dry drill cuttings were placed in a burlap bag and then into a plastic bag. This bag was dropped on top of the plug,, then two shovelfulls of drill cuttings, followed by 14 lbs. (6.4 kg) of Hydromex M210U and 110 lbs. (50 kg) of ANFO. A one pound Procore primer was lowered on primacord, then another 55 lbs. (25 kg) of ANFO poured down the hole. Six feet (1.8m) of drill cuttings were shovelled on top of the column as stemming. (E. Kordas, 1979).

The resulting secondary explosion knocked out a heat probe on the ventilation heaters, so the main ventilation fans had to be shut down before the minus 40*x*F temperatures froze the shaft. The explosion caused the 4-5 stope to upcast to surface through the shaft and through the drill holes (Figure 3.7). Sulphur dioxide polluted the air in the headframe and shifter's offices, where the crews had taken refuge from the stope blast and forced the crews outside to the machine shop. The slow wind present that evening drifted the sulphur dioxide laden air into the machine shop and concentration plant and these places in turn had to be evacuated. The force of the explosion moved a 90 lb. (41 kg) drill rod and demolished a lunchroom near the shaft station (Figure 3.8). The flame front travelled extensively on the 200 level burning hoses on three Cavo loaders and one down- the-hole drill, melting insulation on electrical cables and plastic ropes used as stope fences.

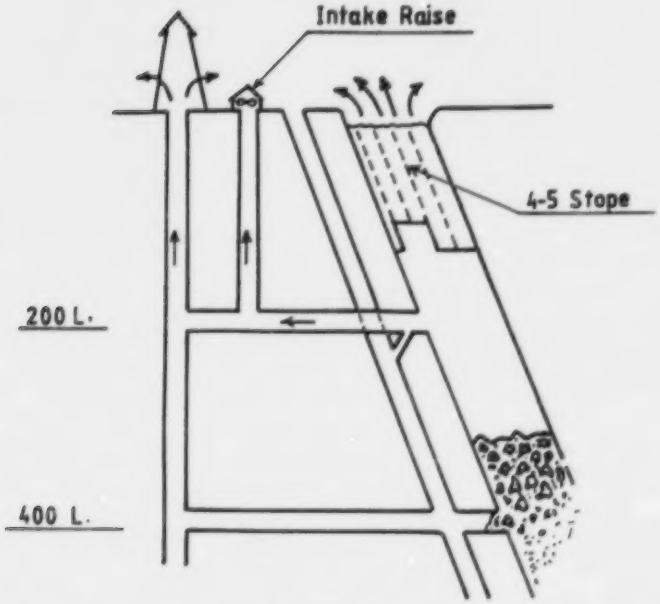


FIGURE 3.7 SECTION THROUGH FOX MINE (After Du Russel)

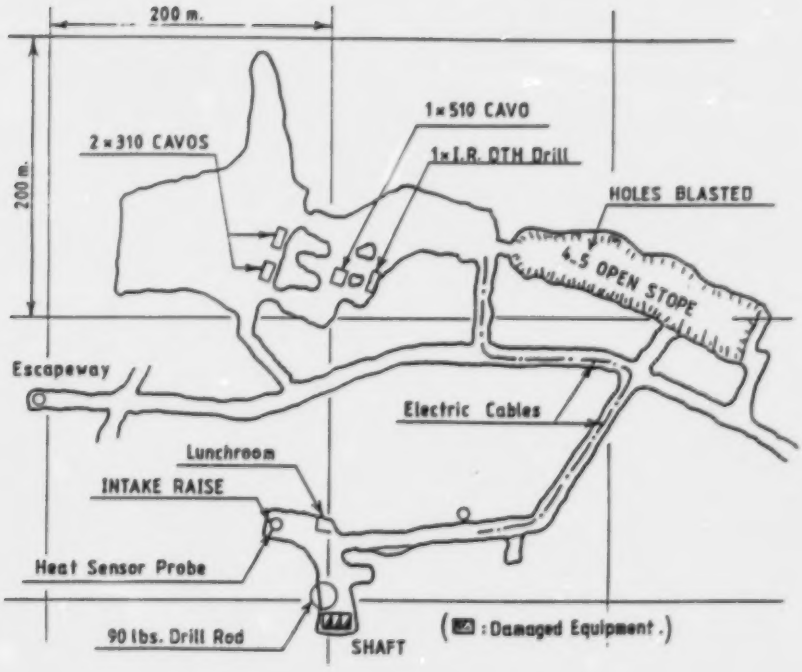


FIGURE 3.8 DAMAGE ON 200 L. FROM 4-5 BLAST FOX MINE
(After Kordas)

3.4 DUST EXPLOSIONS ASSOCIATED WITH DEVELOPMENT MINING

Numerous dust explosions have occurred in drift and raises. In a development end at the Preska Mine in South Africa (Holding, 1975), sulphur dioxide fumes were thrown back a distance of 300m into the main intake where a shift boss was standing. He eventually collapsed and had to receive medical attention.

In May 1966, two men were overcome by sulphur dioxide gas after a drift round was fired on the 600 level of Mines Dumagami Ltée (Scheskus, 1986). The round was drilled in massive pyrite and the level was evacuated before the round was set off. Approximately 10 to 15 minutes later two pipe-fitters returned to the 600 level where they noticed no unusual excessive accumulation of smoke and commenced work. Approximately 10 minutes later while working in the shaft around 40 feet (12m) below 600 level they smelled and tasted gas. They immediately climbed up to the 600 level and called for the cage. While waiting for the cage, acrid yellow smoke smelling of sulphur bellowed into the station. Both men were able to get on the cage and were hoisted to the surface, where they were given oxygen and immediately sent to hospital.

Sulphide dust explosions with resultant minor damage have occurred at mines listed in Table 3.3. Many of the mines have had several dust explosions associated with development work but only the last reported explosion is included in the list.

The sulphide dust explosions which occurred at these mines were comparatively small and caused little damage. However, it should be appreciated that a comparatively small dust explosion can, given the right conditions, lead to development of a major dust explosion in a similar manner in which a small methane explosion at the working face of a

coal mine can lead to the development of a major dust explosion. Many hard rock mines permit blasting in development headings during a shift when the workforce is still underground. Should a minor dust explosion lead to the development of a major explosion then members of the workforce would be at risk.

Table 3.3: Reported Sulphide Dust Explosions Associated with Development Mining.

YEAR	MINES	DEVELOPMENT MINING
1987	Geco, Canada	Raising
1987	Westmin, Canada	Driving
1987	Trout Lake, Canada	Driving
1987	Hellyer, Australia	Raising
1987	Hellyer, Australia	Driving
1987	Elura, Australia	Driving
1986	Woodlawn, Australia	Driving
1986	Dumagami, Canada	Driving
1986	Langsele, Sweden	Driving
1986	Brunswick M & S, Canada	Driving
1985	Lynn Lake, Canada	Driving
1985	Elura, Australia	Raising
1981	Mattabi, Canada	Raising
1981	Mattabi, Canada	Driving
1979	Ruttan, Canada	Driving
1974	Preska, South Africa	Driving
1974	Fox, Canada	Driving

3.5 DUST EXPLOSIONS ASSOCIATED WITH SECONDARY BLASTING

Sulphide dust explosions caused by secondary blasting at drawpoints have been reported by several mines. At the Fox Mine, dust explosions occurred during secondary blasting of drawpoint oversize in two different stopes (Du Russel, 1981). The explosions appeared to consume the dust on the drawpoint floor and walls. The concussion of the secondary blast was not unusual in any noticeable form to the drawpoint operators. However, a long delay period was required to clear sulphide dioxide from the drawpoint. This gas also contaminated the sub-levels above and caused abnormal ventilation flows. The drawpoint floor and walls were covered with 'reddish' remains of burnt dust. The explosive used was Pillowpac.

Green (1986) reported that in the early operating years at Brunswick Mining, Bathurst, and prior to the change to mechanized cut and fill, slusher and drawpoint secondary blasting were the greatest generators of sulphur dioxide. In most cases the ventilation system was not highly developed and problems resulted from recirculation. In secondary blasting, holing was rarely used and sand blasting was the common feature. Attempts made to reduce sulphide explosions included placing small bags of lime next to the charge as well as a small plastic bag of water. Air and water sprays were also installed in the drawpoints and box holes but these were difficult to maintain. As well the washing of the backs and walls of the drawpoints were not always carried out. The hazard from sulphide blasts at secondary blasting sites was ultimately reduced by limiting such blasts to lunchtime and quitting time. Workers were also provided with "sulphur masks" which were frequently used.

Byberg (1986) reported that small sulphur dioxide occurrences have occasionally been experienced, usually after secondary blasting in scrams or boxholes. The few occurrences were relatively minor and were dealt with in a fashion similar to all underground blasts producing blasting gases.

At the Mattabi Mine a sulphide dust explosion occurred in 1981 when 50 lbs. of Amex were used in sand blasting oversize. The dust explosion damaged vent tubing and deposited red dust. At the same mine in 1982, a dust explosion occurred when 12 sticks of powder were used in a pole blast in a drawpoint. No precautions were taken prior to blasting and the blast was initiated at the surface. The secondary explosion damaged bulkheads and displaced air and water lines (J. Wotton, 1986).

4. FACTORS WHICH INFLUENCE THE EXPLOSIBILITY OF A DUST

4.1 INTRODUCTION

The development and severity of a dust explosion are functions of the physical and chemical properties of the dust, concentration of the dust, temperature and energy of the ignition source, oxygen concentration, turbulence of the dust cloud and characteristics of the space enclosing the explosion.

Two of the more important explosion parameters of a dust which may be determined in laboratory or large scale tests are the maximum pressure developed and the rate of pressure rise. In these tests a dust cloud is produced in a closed explosion chamber, the cloud is ignited and the pressure developed in the chamber measured. Explosion chambers vary in volume from 1 litre to 10m^3 and the ignition energy from millijoules to 10 kilojoules. The Hartmann explosion chamber (Figure 4.1) has been used extensively to investigate the explosive characteristics of dusts. A pre-weighed mass of dust is placed in the bottom of the chamber, and an air blast, deflected by the cone at the bottom of the chamber, is used to disperse the dust. The cloud is ignited by an electric arc between the electrodes and the pressure developed in the closed chamber is measured with a pressure transducer connected to a high speed recorder.

The maximum pressure (Pmax) and the rates of pressure rise (dP/dt) are determined from the pressure time record of the explosion as shown in Figure 4.2. The tests are carried out at various dust concentrations, usually between 100 and 3000 g/m^3 to determine the maximum values of Pmax and (dP/dt) for a particular dust. Typical results for such tests on coal, wheat and pyritic dusts are given in Table

4.1. The value of Pmax indicates the maximum pressure that could be expected in a dust explosion and (dP/dt)max the rate at which the explosion develops.

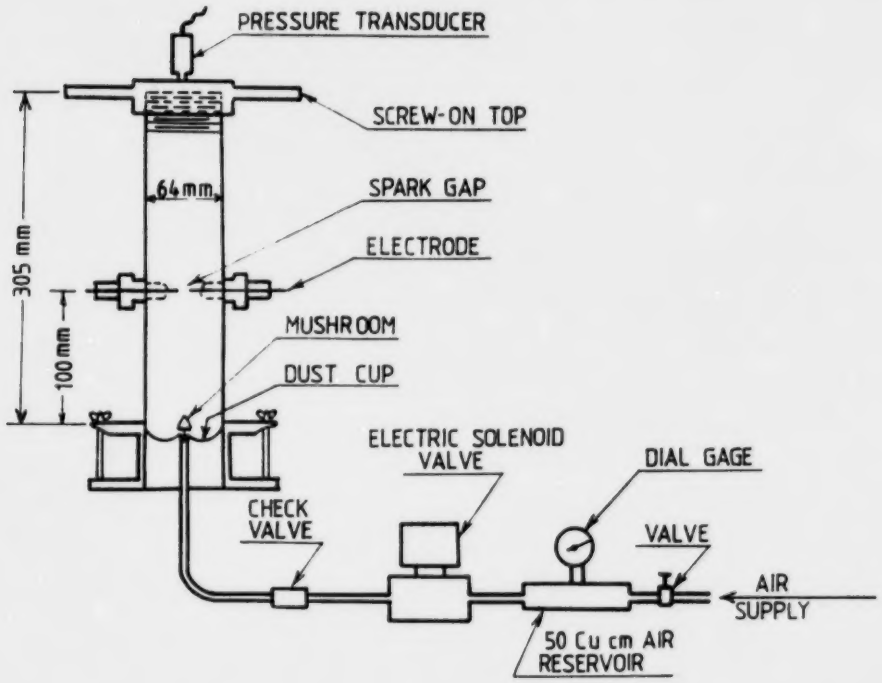


FIGURE 4.1 HARTMANN APPARATUS (1.2 litre)

Table 4.1: Typical Pmax and (dP/dt)max values determined in Hartmann Chamber

DUST SAMPLE	Pmax. kPa	(dP/dt)max kPa/s.
Coal	580	13,000
Wheat	600	11,000
Pyritic ore	500	6,000

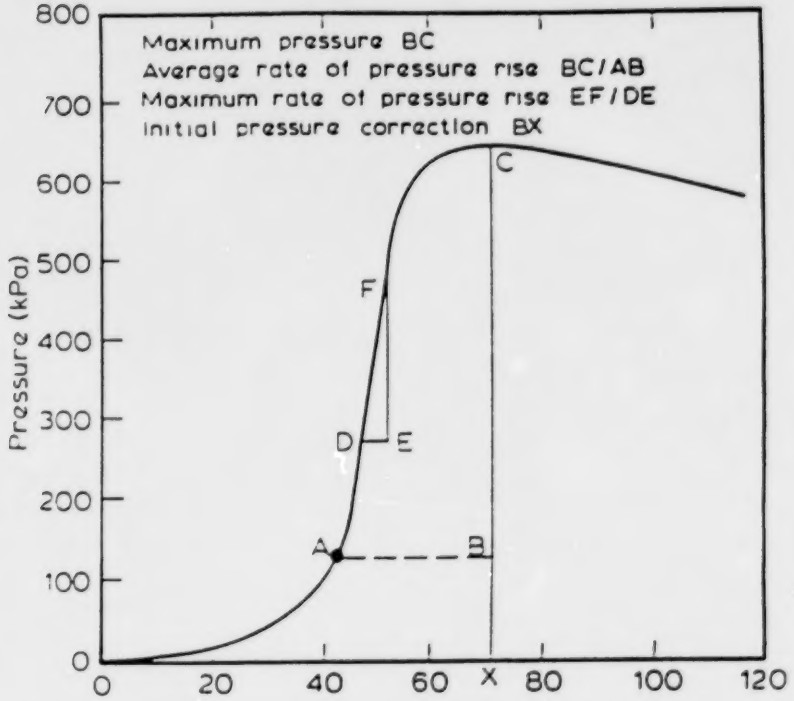


FIGURE 4.2 TYPICAL PRESSURE-TIME CURVE

4.2 PARTICLE SIZE DISTRIBUTION AND SURFACE AREA

A dust is made up of macroscopic solid matter and the particle size, shape, surface condition, porosity and age are variable. A British Standard states that powder is composed of discrete particles of dry material having a maximum dimension of less than 1000 micrometres,'grit' is defined as hard particles, usually mineral, of natural or industrial origin, larger than 75 micrometres while a dust is said to be particulate matter which is, or has been airborne and is less than 75 micrometres in size.

Since combustible particles larger than 75 micrometres are known to be capable of causing an explosion, the term dust, in respect to dust explosions, should be regarded as combustible particles capable of being airborne and generally having a particle size less than 400 micrometres.

The combustion process that occurs during a dust explosion involves chemical reactions at the solid-oxygen interface. Consequently the surface area available for oxidation and heat transfer has a significant effect on the initiation and progress of a dust explosion. When a material is reduced in size the specific surface, i.e. the ratio of the total external and internal surfaces to the total mass, increases and the newly-formed dust is able to burn more easily. In addition, the dust is easier to disperse and maintain in suspension. The increase in surface area with a reduction in particle size is illustrated in Figure 4.3.

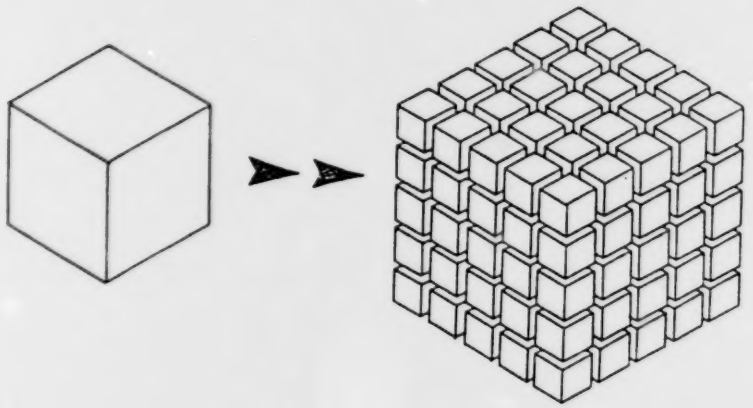


FIGURE 4.3 ILLUSTRATING INCREASE IN SURFACE AREA WITH REDUCTION IN PARTICLE SIZE

The effect of particle size on the maximum rate of pressure rise and the maximum explosion pressure is shown in Figure 4.4. In general, dusts with a particle size of the order of 400 micrometres cannot be caused to explode even with high ignition energies (Bartknecht, 1981). A decrease in particle size generally results in a lower ignition temperature, with a limiting ignition temperature approached at around 100 micrometres (Nagy and Verakis, 1983). If surface area were not a factor, the ignition temperature would be independent of the particle size.

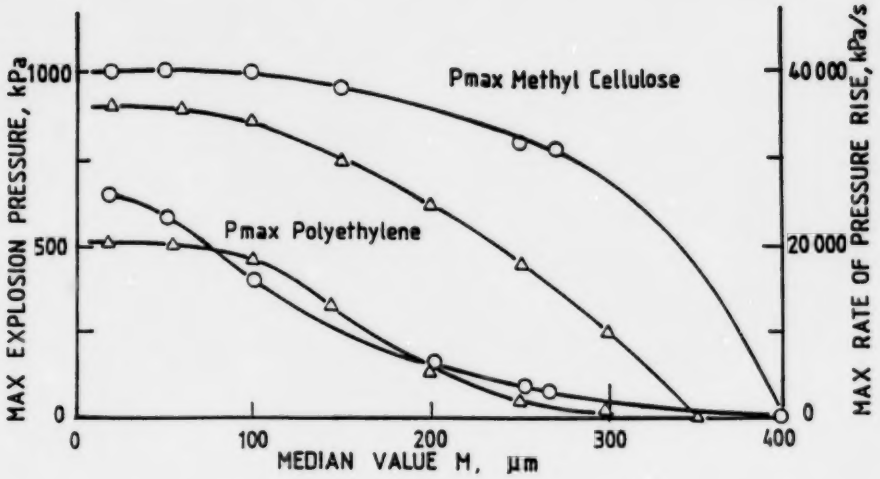


FIGURE 4.4 INFLUENCE OF PARTICLE SIZE ON THE EXPLOSION DATA (After Bartknecht, 1981)

4.3 PARTICLE DENSITY

The ease with which dust particles are dispersed and the time required for a dust to settle out are functions of particle size, shape and density. Settling velocities, in air, for various spherical particles are given in Table 4.2 (Perry, 1963).

Table 4.2: Settling Velocities in air of Spherical Particles

SIZE	Settling Velocity m/s		
	Pyrite	Limestone	Coal
400	5.48	3.96	2.13
100	1.07	0.61	0.37
50	0.37	0.24	0.12

4.4 DUST CONCENTRATION

The combustion reaction of a gas-air mixture involves a homogeneous mixture of molecular-sized particles and the upper and lower explosive concentrations are well defined. The upper and lower explosive limits of an air-dust mixture are difficult to determine due to problems associated with generating a uniform dust cloud. The lower explosion limit, i.e. the minimum explosive concentration, is the minimum dust concentration that is capable of igniting and propagating flame. At dust concentrations below the minimum value the particles are separated by comparatively large distances and the heat liberated by the oxidation of a single particle is not sufficient to ignite adjacent particles. Above the upper explosive limit the particles are closely packed and prevent the ready access of oxygen required in the combustion process. Between the lower and upper explosion limits the dust concentration affects the severity of the explosion.

For dust explosions the maximum explosion pressure Pmax and the maximum rate of pressure rise (dP/dt)max, are determined by test explosions over a wide range of concentrations in an explosion chamber. The explosion parameters versus dust concentration determined for an ore containing pyrite and having a 41% sulphur content are shown in Figure 4.5. The maximum pressure developed and maximum rate of pressure rise in an 8 litre chamber occurred at a dust concentration of around 1200 g/m^3 compared with the stoichiometric concentration of 400 g/m^3. Bartknecht (1981) observed that for many combustible dusts the optimum explosion effects were observed at concentrations two to three times higher than the stoichiometric composition of the dust-air mixture. Theoretically, the most serious explosion effects might be expected with a stoichiometric mixture. However, the explosion of a dust-air mixture normally leaves a proportion of partially consumed material.

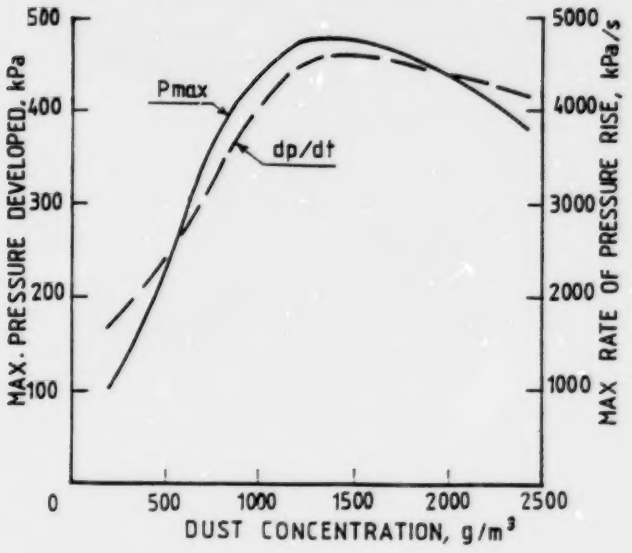


FIGURE 4.5 INFLUENCE OF DUST CONCENTRATION ON EXPLOSION DATA. (After Enright, 1985)

4.5 CHEMICAL COMPOSITION AND ADMIXED INERT DUSTS

As a dust explosion propagates a chemical reaction occurs between the dust particles and the atmospheric oxygen, the rate at which oxygen is consumed being directly related to the chemical nature of the dust. When pyrite is exploded the products of combustion are usually; magnetite, Fe_3O_4 or hematite, Fe_2O_3, and sulphur dioxide.

Typical reactions are:

$$3FeS_2 + 8O_2 = Fe_3O_4 + 6SO_2 + 2462 \text{ kJ/mol.}$$
$$4FeS_2 + 11O_2 = 2Fe_2O_3 + 8SO_2 + 3458 \text{ kJ/mol.}$$

The maximum explosion pressure and maximum rate of pressure rise decrease as the pyrite content of an ore decreases. Ores containing less than 40 percent pyrite are difficult to explode in test chambers with an ignition source of 20,000 J (Enright, 1985). The addition of inert dust inhibits the

development of a dust explosion by several different methods. (Nagy and Verakis, 1983, Richmond et. al., 1975). The combustion of the dust is retarded as a portion of the heat of combustion is transferred to the inert dust. This dust also inhibits the diffusion of oxygen to the burning particles and restricts diffusion of gases from the particles. In addition, the inert particles shield the combustible dust from radiation energy during the combustion reaction. The effects of the addition of limestone on the explosion parameters of an ore containing around 80 percent pyrite are shown in Figure 4.6. The curves show that the addition of up to 20 percent limestone does not result in a significant decrease in the maximum pressures developed although higher dust concentrations are required to obtain the same pressure rise as with ore. (Enright, 1985).

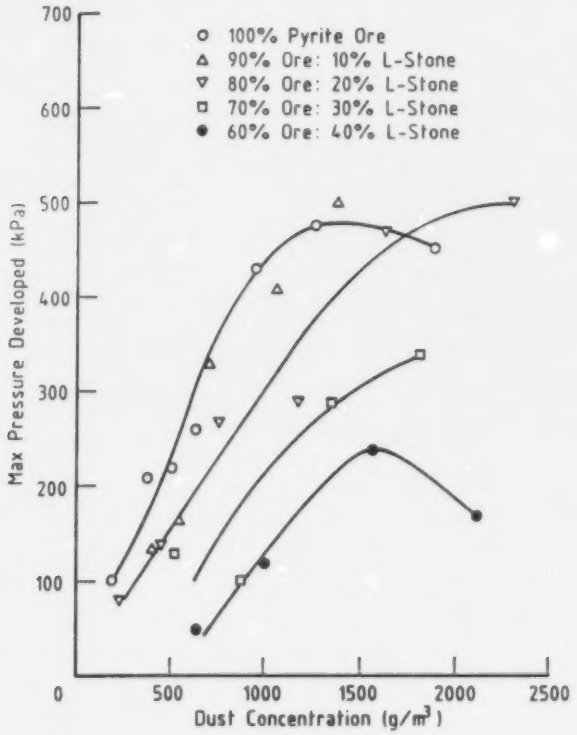


FIGURE 4.6 PMAX VERSUS DUST CONCENTRATION WITH ORE-LIMESTONE DUST MIXTURES (Enright, 1985)

4.6 OXYGEN CONCENTRATION

A dust explosion requires not only an explosive concentration of dust and an ignition source, but also sufficient concentration of oxygen to support combustion. Nagy and Verakis (1983) reported that in a series of experiments in a 1.2 litre Hartmann explosion chamber with coal and aluminium dusts that a slight reduction in oxygen concentration from that of a normal atmosphere produced a moderate decrease in explosion pressure but as the limiting oxygen concentration was approached the explosion pressure decreased rapidly (Figure 4.7).

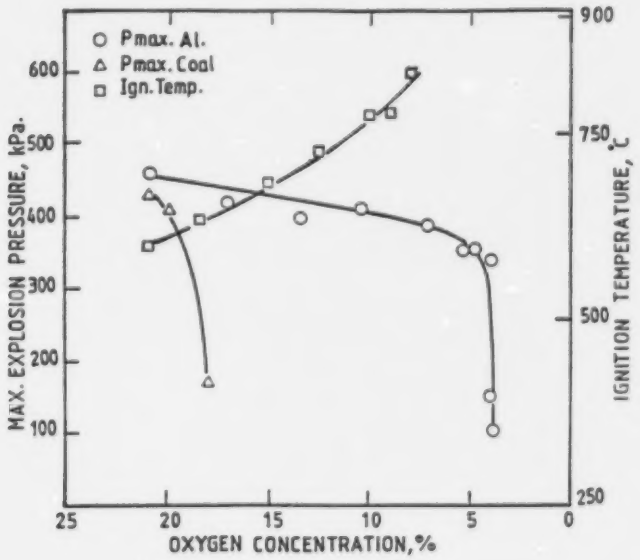


FIGURE 4.7 EFFECT OF OXYGEN CONCENTRATION ON EXPLOSIBILITY DATA. (After Nagy & Verakis, 1983)

A reduction in oxygen content also decreases the probability of a dust explosion as the ignition temperature of a dust dispersion increases as the oxygen concentration in the atmosphere decreases (Nagy and Verakis, 1983; Craven and Foster, 1967).

4.7 IGNITION ENERGY AND TEMPERATURE

Ignition of dusts may be caused by frictional spark, electric spark, flame, heated surface or by spontaneous heating. These igniting sources are characterised by energy, power and temperature. A frictional or electrical spark is usually expressed in terms of minimum energy to ignite a dust cloud whereas a flame or heated surface is characterised by temperature.

Most dust explosions occur when air movement or air blast disperses dust onto an ignition source. Jacobson et. al. (1964) found the minimum ignition temperature for minus 75 micrometre pyrite particles in air was 380°C and Sychev et. al. (1975) reported a minimum ignition temperature of 300°C for minus 10 micrometre particles. In general, a decrease in particle size results in a lower ignition temperature and a decrease in oxygen content results in an increase in ignition temperature.

4.8 ADDITION OF MOISTURE

The ignition temperature of a dust cloud increases as the moisture content of the dust increases (Nagy and Verakis, 1983).

Tests by Enright (1985) with two coal dusts in 8 and 20 litre explosion chambers showed that the explosion pressure decreased as the moisture content of the coal was increased, as shown in Figure 4.8. The initial rate of decrease was low but as a limiting moisture content was approached the rate of decrease became higher. This rapid decrease was not considered to be primarily due to the effects of water but rather to the difficulty in dispersing the high moisture dusts in the explosion chambers.

Cybulski (1975) found that coal containing water in excess of its absorbing power propagated an explosion of tremendous violence when the initiator was sufficiently strong. Eckholff and Mathisen (1978) reported that tests by Scholl in a 1m³ vessel revealed that peat dust with a moisture content around 50 percent was capable of producing an explosion.

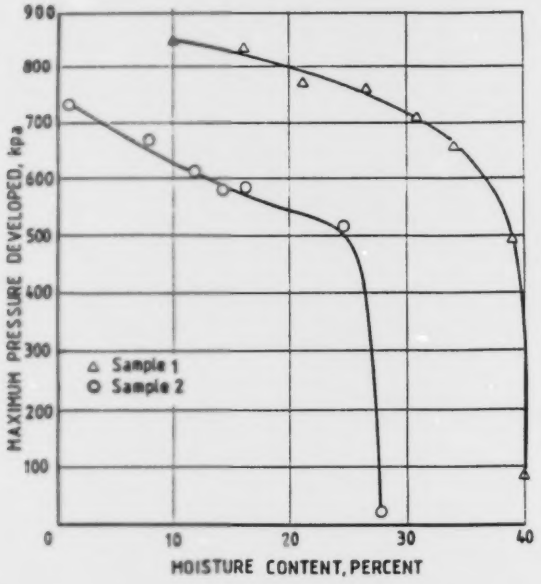


FIGURE 4.8 PRESSURE RISE VERSUS MOISTURE CONTENT
(Enright, 1983)

4.9 TURBULENCE

An initial degree of turbulence is required for the development of a dust explosion and the degree of turbulence may be increased as the pressure wave and flame front move through the underground workings. Increased turbulence will result in an increased explosion hazard as access of oxygen to the dust surface is increased and the fragmented flame front provides new sites from which combustion can develop simultaneously. Bartknecht (1981) showed that increased turbulence resulted in a marked increase in the rate of pressure rise and higher explosion pressures which tended towards a limiting value.

5. SULPHIDE DUST EXPLOSIONS

5.1 INTRODUCTION

In a hard rock mine the generation and ignition of the dust cloud are usually associated with the use of high explosives in stoping operations, development work and secondary blasting. Dust clouds may also be formed during the breakage and subsequent movement of the rock. Detonation of explosives during and after the formulation of the dust cloud may provide ignition sources and cause a primary dust explosion. The pressure wave of a primary explosion may disturb dust on the roof, walls and floor of the workings and ignition of this dust by the primary flame front will result in a violent, self-sustaining secondary dust explosion. Figure 5.1 shows a self-sustaining dust explosion in a development heading.

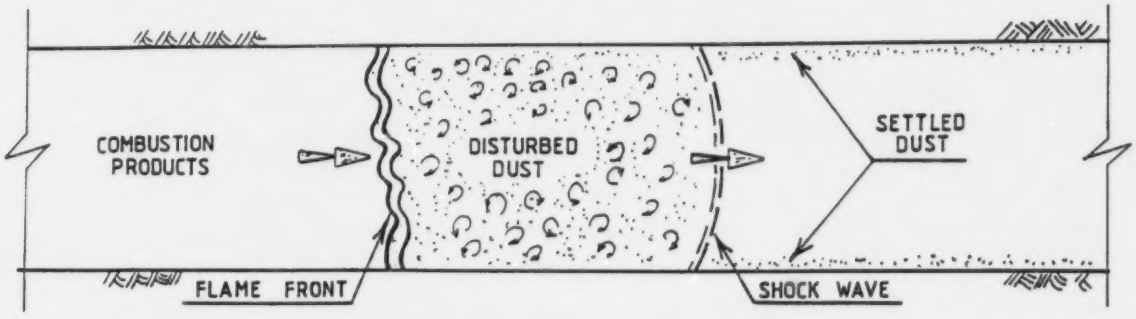


FIGURE 5.1 SELF SUSTAINING DUST EXPLOSION

5.2 SULPHIDE MINERALS

The majority of dust explosions in metalliferous mines have been associated with mining operations in massive pyritic or pyrrhotitic orebodies. Laboratory tests have shown that sphalerite dust is capable of sustaining an explosion and it is possible that other minerals with an appreciable sulphur content may form an explosive dust cloud. The sulphur content of selected minerals is given in Table 5.1.

TABLE 5.1: Sulphur Content of Selected Minerals

MINERAL	FORMULA	SULPHUR %
Pyrite	FeS_2	53.3
Pyrrhotite	Fe_{1-x} S	50-55
Chalcopyrite	$CuFeS_2$	34.9
Sphalerite	ZnS	32.9
Arsenopyrite	Fe A_sS	19.6
Galena	PbS	13.4

Pyrite is by far the most widespread and commonly occurring sulphide mineral. It can be found in almost any type of geological environment: as an accessory mineral in igneous rocks, as a magmatic segregation from igneous rocks, in contact metamorphic deposits and in sedimentary and metamorphic rocks. Pyrrhotite occurs principally in basic igneous rocks, either disseminated or segregated in large masses. Chalcopyrite, pentlandite and other nickel sulphides are commonly associated with pyrrhotite. Relevant properties of pyrite, pyrrhotite and chalcopyrite are given in Table 5.2.

Table 5.2: Properties of Pyrite, Pyrrhotite and Chalcopyrite

	PYRITE	PYRRHOTITE	CHALCOPYRITE
Formula	FeS_2	Fe_{1-x} S	$CuFeS_2$
Sulphur %	53.3	50-55	34.9
Fracture	conchoidal	uneven	uneven
Relative density	5.0	4.6 - 4.7	4.1 - 4.3
Hardness (Mohs')	6 - 6.5	3.5 - 4.5	3.5 - 4.0
Luster	metallic	metallic	metallic
Magnetism	non-magnetic	magnetic variable	non-magnetic

5.2 CHEMICAL REACTIONS

When a dust explosion propagates, complex chemical reactions occur between the dust particles and the atmospheric oxygen. One or more of the following reactions may occur during a pyritic dust explosion:

$$FeS_2 + O_2 = FeS + SO_2 + 334 \text{ kJ/mol}$$
$$2FeS_2 + 5O_2 = 2FeO + 4SO_2 + 1435 \text{ kJ/mol}$$
$$3FeS_2 + 8O_2 = Fe_3O_4 + 6SO_2 + 2462 \text{ kJ/mol}$$
$$4FeS_2 + 11O_2 = 2Fe_2O_3 + 8SO_2 + 3458 \text{ kJ/mol}$$

The reaction products may include ferrous sulphide (FeS), ferrous oxide (FeO), ferroso-ferric oxides (Fe_3O_4) and ferric oxide (Fe_2O_3) and SO_2. Finely divided ferric oxide obtained by burning pyrite is used as a red pigment or a mild abrasive under the name of Venetial Red or rouge. As the mineral hematite, ferric oxide occurs in large quantities in the earth's crust. When ferric oxide is heated above 1000°C ferroso ferric oxide, Fe_3O_4, is obtained (Sisler, 1957). This compound occurs naturally as mineral called magnetite, some samples of which are highly paramagnetic. Magnetite may be regarded as a compound of ferrous and ferric oxides with the formula written as $FeO.\ Fe_2O_3$.

Ferrous sulphide and sulphur vapour are formed during the thermal decomposition of pyrite

$$FeS_2 = FeS + S - 78 \text{ kJ/mol}$$

Gardner et al (1926) suggested that hydrogen sulphide was formed by the reaction of sulphur vapour with water

$$3\ S_2 + 4H_2O = 4\ H_2S + 2\ SO_2$$

5.3 SULPHUR CONTENT OF DUST

The explosion parameters, P_{max} and $(dP/dt)_{max}$, of six sulphide ores were determined in 8 and 36 litre explosion chambers. Two samples were comparatively pure pyrite and the other samples were from mines in New South Wales and Canada. Analyses of the ore samples are given in Table 5.3.

TABLE 5.3: Ore Sample Analysis

SAMPLE	SULPHUR	IRON	LEAD	ZINC	OTHER
Geological	52.0%	46.3%	930ppm	1800ppm	1.7%
Geological	47.1%	47.1%	1080ppm	832ppm	5.8%
Elura	41.4%	40.7%	3.0%	6.7%	8.2%
Hellyer	40.9%	30.1%	7.0%	13.0%	9.0%
Timmins	45.9%	39.9%	0.3%	0.8%	13.1%
Sullivan	31.5%	48.7%	2.4%	5.5%	11.9%

Various dust-limestone mixtures were also tested in the chambers to determine the effects of a reduction in sulphur content of the ores. In Figures 5.2 and 5.3 the explosion parameters are plotted against the total sulphur content of ores. Total sulphur content rather than the pyritic sulphur content of the dust was selected as it was difficult to estimate to what degree metallic sulphides other than pyrite, present in the ore contributed to the development of an explosion. The data presented in Figures 5.2 and 5.3 were obtained when dusts with a contained concentration of 1500 g/m^3 of ore were exploded in the chambers. The total dust concentration of the ore-limestone mixtures dispersed in the chambers increased in proportion to the mass of limestone in the mixture.

The results shown in Figure 5.2 indicate that the initial decrease in explosion pressure with a reduction in sulphur content is slight, but as a limiting value is

approached, the rate of decrease in pressure becomes higher. No explosions were obtained with dusts containing less than 18 percent total sulphur and an ignition source of 1.5g nitrocellulose. At high sulphur contents the maximum rate of pressure rise decreased almost linearly with an increase in inert material.

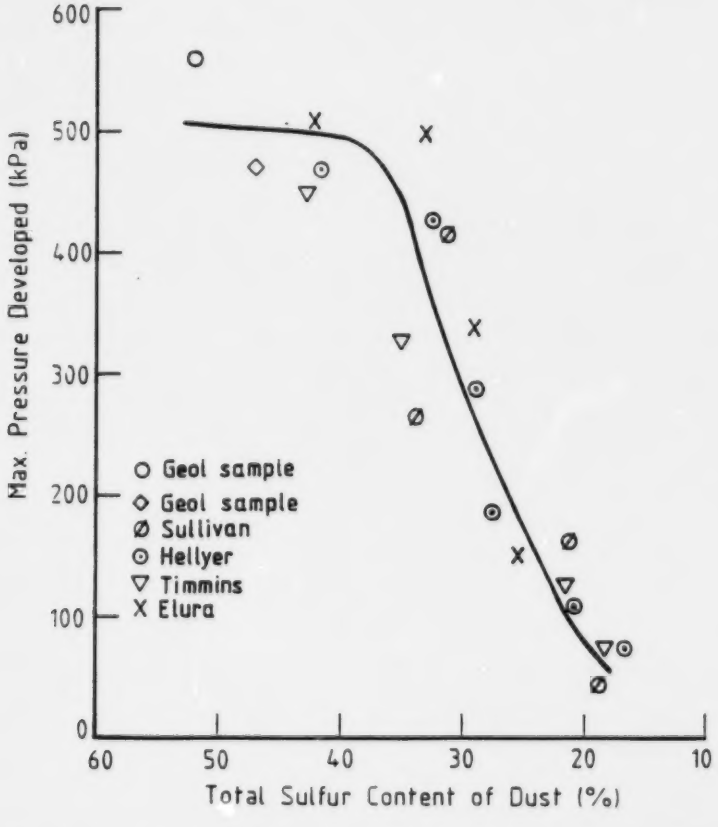


FIGURE 5.2 P_{max} VERSUS TOTAL SULPHUR CONTENT OF DUST

5.4 MINIMUM EXPLOSIVE CONCENTRATION

Tests in an 8 litre chamber have shown that a pyritic ore (37.3% S) and pyrrhotitic ore (33.5% S) were each capable of producing significant explosion pressures and rates of pressure rise at dust concentrations of 190 g/m^3 (Enright, 1984). These concentrations do not represent the lowest explosive concentrations as the lower explosive limit

of a dust is a function of the method of dust dispersion, strength of ignition source and the size and shape of the explosion chamber (Hertzberg et al., 1979).

A very thin layer of dust on the surfaces of a drive or stope will provide sufficient mass of dust to form an explosive concentration. For example, in a 5m x 3m drive a dust layer 0.6mm in thickness on the roof, would produce a dust concentration of 500 g/m³ when disturbed. In a stope void with dimensions of 25 x 25 x 50m a layer of dust 1.0mm thick could produce a dust cloud with a concentration of 500 g/m³.

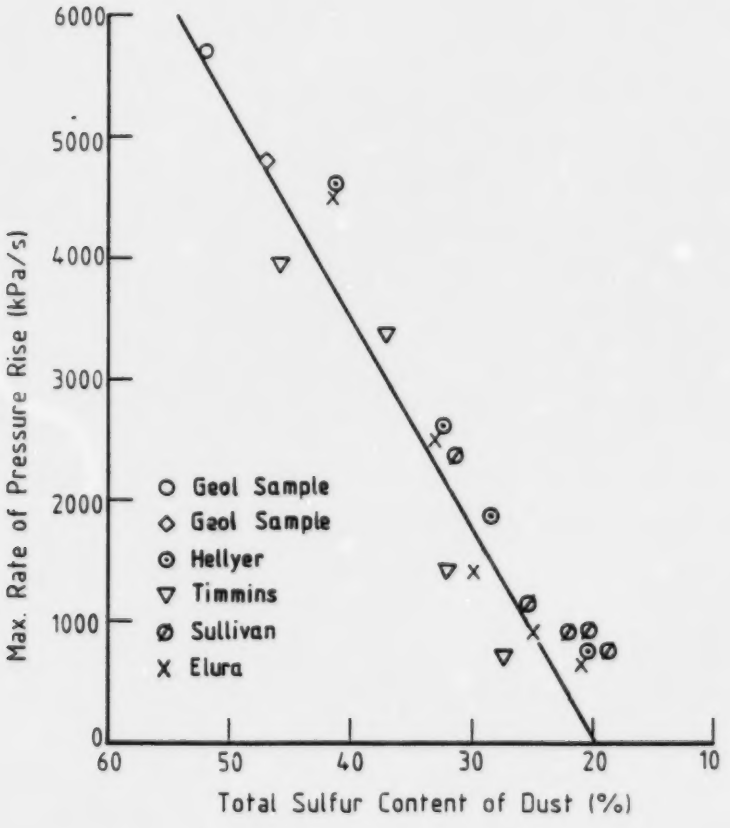


FIGURE 5.3 $(dP/dt)_{max}$ VERSUS TOTAL SULPHUR CONTENT OF DUST

5.5 PARTICLE SIZE DISTRIBUTION

Dusts with a particle size above 400 micrometres cannot be caused to explode even with high ignition energies, (Section 4.2), and the explosibility of a dust increases with a decrease in particle size as shown in Figure 4.4. In the workings of a sulphide orebody it is possible that pyrite or pyrrhotite may tend to produce more fine material when blasted or crushed, than the gangue minerals and that the sulphur content of the fine dust on the surface of the workings may be higher than the average for the orebody. The tendency of some minerals to produce "fines" may explain why dust explosions have occurred in mines working a comparatively low sulphur content ore.

5.6 SULPHIDE DUST EXPLOSIONS IN OPEN STOPES

GENERATION OF DUST CLOUDS

The generation and ignition of dust clouds in open stopes are usually associated with the use of high explosives to blast rock from the face. Two sources of fuel for the explosion are the dust previously deposited on the surfaces in the stope and dust produced during blasting. The dust deposited in the stope prior to the blast may be the result of previous blasts, dust formed during movement of the rock in the stope to the drawpoint, and dust produced during loading at the drawpoint. Dust produced during loading may be transported by the ventilating air into the stope and deposited in the stope.

During the stope blast, dust is generated and clouds formed by the following processes:

1. Detonation of a column of explosives creates strong compressive stress waves that travel away from the explosion, and are later reflected as tension waves

from the free face. In the immediate vicinity of each charge, the radial strain exceeds the rock's dynamic breaking strain and an annular volume of rock is crushed. Immediately outside the crushed zone, the tangential strains create a radical crack pattern as shown in Figure 5.4. Little data has been published on the mass of dust produced during blasting. Langefors and Kihlstrom (1967) estimated that the thickness of the crushed zone, shown in Figure 5.4 was of the same order as the radius of the hole. Gustafsson estimated that the diameter of the pulverized section around a 100mm diameter hole was 150mm and that beyond this zone the rock was crushed and not completely pulverized. Not all the dust in the pulverized zone can contribute to an explosion, as dust with a particle diameter greater than 400 micrometres cannot explode. During blasting the ore adjacent to the explosive column would be subjected to high temperatures and pressures, and chemical changes may occur. However, the speed of the explosive reaction and the insulating properties of the rock would limit this affected zone.

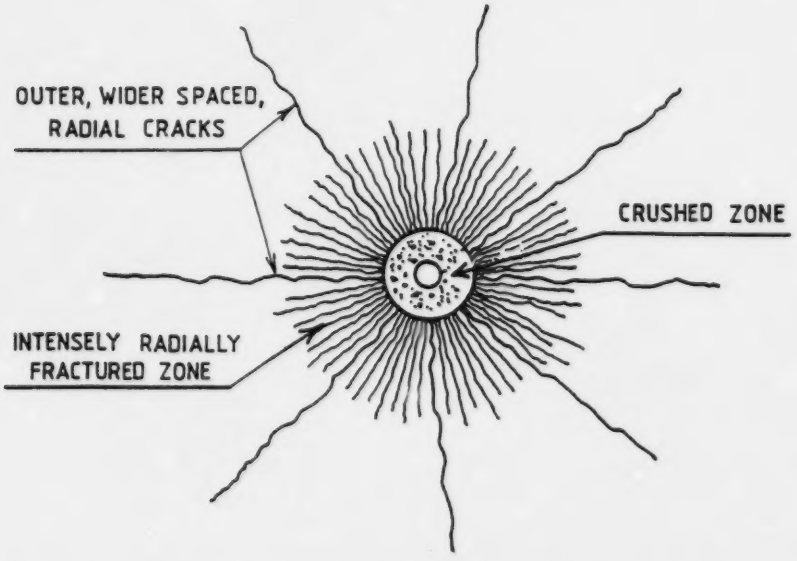


FIGURE 5.4 CRUSHED AND FRACTURED ZONES AROUND BLAST HOLE

2. When primary cracks have been created from the face to the blast hole, a secondary fragmentation takes place through gas penetration and the presence of a semistatic stress field. When the fragmentation front reaches the face the fully broken rock bursts out. During this mechanism gas penetration into the stemming takes place and pulverized dust may be ejected from the blast hole (Favreau et al., 1986).

3. Dust is created by in-flight collisions of the broken rock and when rock strikes the walls and floor of the stope. Evidence of this dust cloud is observed when blasted rock strikes the floor of an open-cut mine.

4. Surface waves, gas venting from the blast holes and air movement in the stope disturb dust lying on the roof, walls and floor of the stope.

5. Vibrations caused by blasting in nearby workings may disturb previously deposited dust in the stope so that a cloud is generated prior to blasting in the stope.

6. The pressure wave and flame front associated with a primary dust explosion may cause dust previously deposited to provide fuel for a self-sustaining secondary explosion.

DUST IGNITION

For a dust explosion to occur the airborne dust must be exposed to an ignition source of sufficient temperature and duration to ignite the dust. Detonation of explosives during and after the formation of the cloud or the burning of undetonated explosives may provide ignition sources. Frictional ignition by pyritic ore is another possible, though less likely, ignition source. Misfires may lead to

conditions where explosives may be ignited or detonated after the normal blast. The techniques developed to reduce blast vibrations, such as deck charging, air spacers, and vertical crater retreat mining may lead inadvertently to an increase in the risks of a dust explosion.

OXYGEN CONCENTRATION

Detonation of high explosives produces a large volume of inert gases and these gases may displace air from the stope. One kilogram of ANFO generates around 1.3m^3 of explosion gases with the following analysis:

Carbon monoxide	0.8%
Carbon dioxide	9.0%
Hydrogen	1.7%
Water	61.7%
Nitrogen	26.8%

A reduction in the oxygen content of the atmosphere may prevent or inhibit the development of a dust explosion.

5.7 MODEL OF A STOPE BLAST

ROCK DISPLACEMENT IN A STOPE

A study of the sequence of the events which take place during and after a stope blast may enable current techniques to prevent dust explosions to be reviewed and indicate new approaches to the problem. The BLASPA (Favreau, 1983, 1986) simulation model was used to study rock movement and gaseous conditions in a stope with ANFO and non-aluminised slurry as blasting agents. The stope was considered to be partially mined out, and a section through the stope is shown in Figure 5.5. The BLASPA model gave rock velocities and displacements at 10m vertical and 1m horizontal intervals. Design parameters for the ANFO blast were as follows:

Young's Modulus	122 x 10^3 MPa
Poisson's Ratio	0.26
Rock density	3.8 t/m^3
Hole diameter	0.2m
Hole depth	50m
Burden and spacing	4m x 4m
Top stemming	9m
Bottom stemming	1m
Explosive density (ANFO)	850 kg/m^3
Column height (ANFO)	40m

Velocities of the rock mass at the face and adjacent to the blast hole were determined between 25 and 800ms using the BLASPA model. It was assumed that these velocities were horizontal and gravity was the only influence on the vertical movement of the rock. After an elapsed time of 150ms the rock velocities at the centre of the face were approximately constant at 27 m/s with ANFO and 20 m/s with slurry explosives. Vertical movement of the rock can be calculated with the following equation:

$$s = ut + 0.5\,gt^2$$

where s = vertical displacement (m)

u = initial velocity (m/s)

g = acceleration due to gravity (9.8 m/s^2)

t = elapsed time (s)

The horizontal and vertical rock displacements, following the detonation of one hole loaded with ANFO, are given in Table 5.4.

Table 5.4: Horizontal and Vertical Displacement of Blasted Rock

ELAPSED TIME (ms)	HORIZ. DISP. (m)	VERT. DISP. (m)	COMMENTS
100	2.7	0.05	
200	5.4	0.2	
740	20.0	2.7	initial collision with walls
1430	20.0	10.0	initial collision with draw cone
3500	20.0	60.0	final collision with draw cone

Profiles of the moving rock mass at 250 and 750 ms after detonation of one hole loaded with ANFO are shown in Figure 5.5. Similar profiles were obtained with a hole loaded with slurry explosive.

DUST CLOUD GENERATION

The model indicated that four separate mechanisms may be involved in the generation of dust clouds:

1. Detonating cord may disturb dust in the vicinity of the blast holes.

2. Around 100 ms after detonation of the explosives the air movement and surface waves disturb dust previously deposited in the stope.

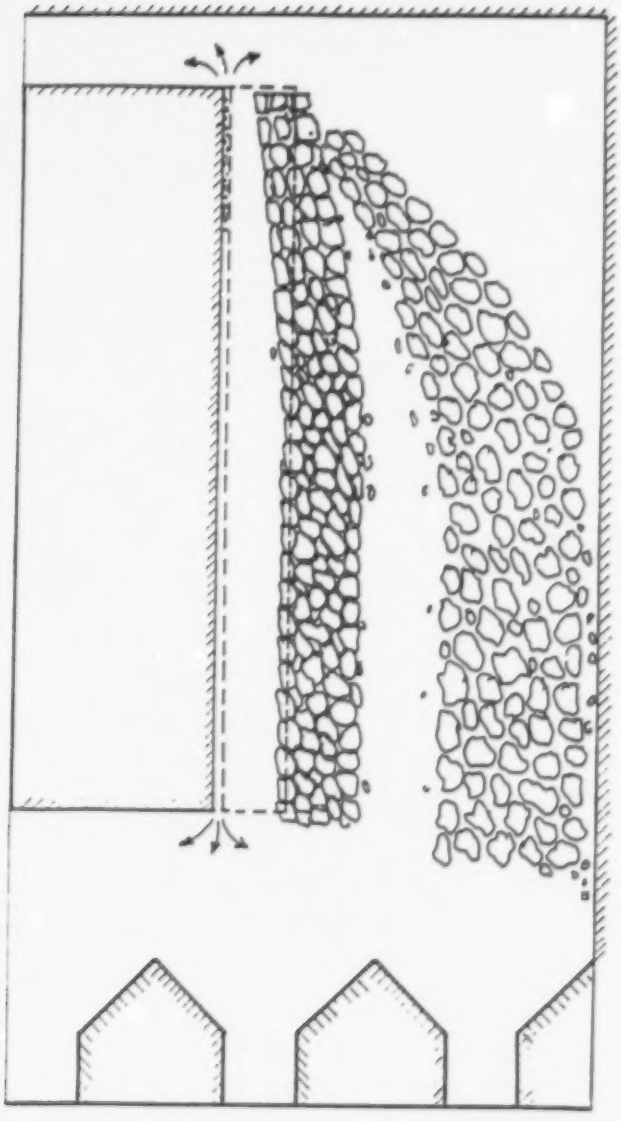


FIGURE 5.5 BLASTED ROCK PROFILES OF ONE HOLE FIRED AT 250 AND 750 ms

3. Gases venting through the blast hole, either due to inadequate stemming or channeling in the stemming caused by the use of detonation cord, carry dust from the pulverized zone of the blast hole. The vent gases also disturb previously deposited dust.

4. A dust cloud is generated when the broken rock hits the walls and bottom of the stope. Formation of the cloud will continue until all the broken rock has come to rest in the bottom of the stope. This particular mechanism would form a dense dust cloud up to 3.5 seconds after detonation in the stope under evaluation.

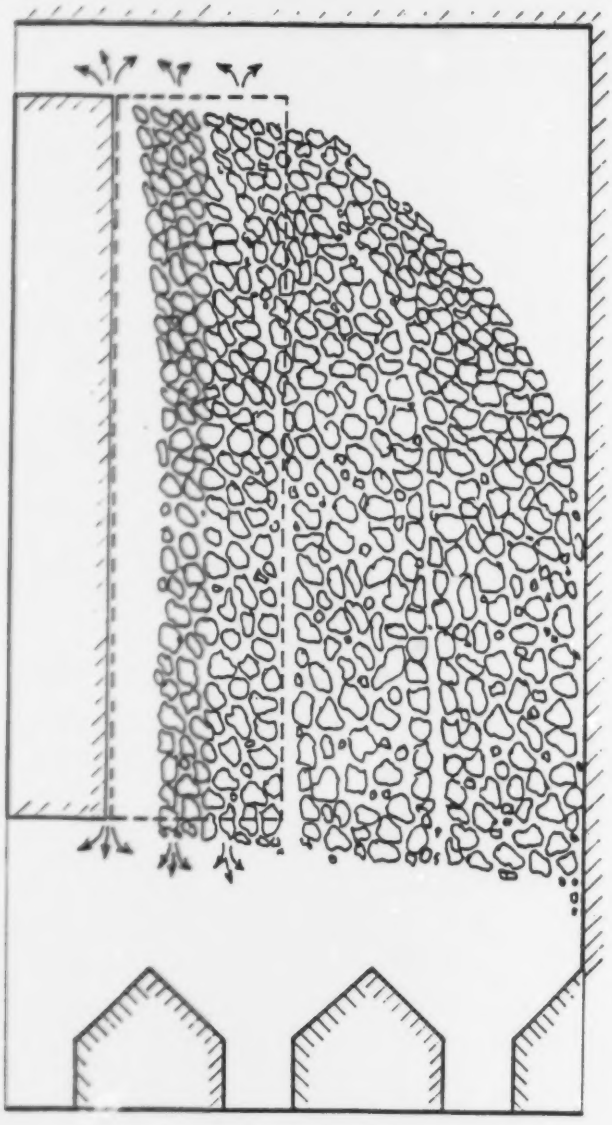


FIGURE 5.6 BLASTED ROCK PROFILE AT 750 ms WITH 3 ROWS FIRED

DUST IGNITION IN STOPE MODEL

The potential ignition sources in the stope are hot gases venting from the blast holes, burning of undetonated explosives and packaging, and frictional ignition as rock masses collide with the walls and floor of the stope. Other concurrent blasting operations, such as stripping a drill drive in a stope or secondary blasting at drawpoints, may lead to dust ignition.

The temperature and pressure of the explosion gases can be determined using the BLASPA model if it is assumed that stemming prevents venting of the gases through the top and bottom of the holes and the gases are confined between the new face and the moving rock mass. The gas temperature at various times is given in Table 5.5.

Table 5.5: Estimated Gas Temperatures

TIME ms	VOLUME OF SPACE BETWEEN BLASTED ROCK AND NEW FACE, m^3	TEMPERATURE °K
50	120	760
100	280	560
150	430	471
250	760	400

At 150 ms after detonation, the gas temperature has decreased from around 3000°K to 471°K which is below the minimum ignition temperature of 573°K for 10 micron diameter pyrite particles reported by Sychev et. al. (1978). The low oxygen concentration in the space between the new face and the moving rock mass will inhibit ignition for the dust for a significant time after detonation.

OXYGEN CONCENTRATION IN STOPE MODEL

In the model under consideration, each hole was loaded with 1070 kg of ANFO which produced 1280m³ at 334°K of inert gaseous products. If it is assumed that this gas mixture displaces an equal volume of air from the stope, then the composition of the resultant gas mixture can be calculated for various blast patterns. A summary of these calculations is given in Table 5.6.

Table 5.6: Resultant Oxygen Concentration in Stope for Various Blast Patterns

NO. OF HOLES	OXYGEN CONCENTRATION %
1	19.4
6	16.2
12	12.3
18	8.6

The effect of reduced oxygen concentration on the explosion parameters of coal dust is shown in Figure 4.7. The ignition temperature of pyritic dust increases with a reduction in oxygen concentration and it is unlikely that a dust explosion will occur when the oxygen concentration is less than 15 percent. The oxygen concentration in the stope was reduced to approximately 19 percent when one hole was fired and 12 percent when two rows of six holes were fired. It appears that, under certain conditions, the possibility of a dust explosion may be greater when a small number of holes are fired in the stope.

5.8 DUST EXPLOSIONS IN DEVELOPMENT HEADINGS

DUST EXPLOSIONS IN DRIVES

The development of a dust explosion in a drive is somewhat similar to the development of a coal dust explosion. When the explosives in the cut hole are detonated a stream of hot gases and pulverised ore are ejected from the face. The hot dust is initially contained in an envelope of hot inert gases but inertial and gravitational influences will cause the dust to enter air surrounding the gas envelope. In addition, the hot gases and dust will be dispersed in the stationary atmosphere near the face. The hot pyritic dust may ignite spontaneously when it enters an atmosphere containing sufficient oxygen or dust ignition may be caused by hot explosion gases or burning particles of explosive packaging.

Ignition of the dust particles ejected from the cut holes may result in a primary dust explosion. The air blast caused by detonation of explosives in the cut holes will disturb dust lying on the surfaces of the drive and ignition of this dust may result in a secondary explosion.

MODEL OF A DEVELOPMENT BLAST

A typical cross-section and blast hole pattern for a 2.7m x 2m drive are shown in Figure 5.7. The 33 blast holes 50mm diameter and 3 reamed cut holes 90mm diameter are drilled for a length of 3m. A mass of 5 kg of ANFO required to load each hole to the collar will produce around 6m^3 of inert explosion gases. When NONEL long delay detonators are used then the volume of inert gas produced when sections of the face, as shown in Figure 5.7, are fired is as follows:

No. of Holes Fired	NONEL Delay Numbers	Elapsed Time (s)	Cumulative volume of gas (m^3 at N.T.P.)
3	1,2,3	0.8	18
3	4,5,6	2.0	36
4	7,8,9,10	3.4	60
7	11,12,13	4.9	102
6	14	5.4	138
4	15	6.6	162
6	16	7.3	198

Dust particles in suspension near the face can be ignited but the probability of dust ignition would decrease as the oxygen content of the atmosphere near the face decreased. If it is assumed that the inert explosion gases displace an equal volume of air, then the oxygen content in 20m of drive next to the face would be reduced to around 16 percent after the first 6 holes were fired.

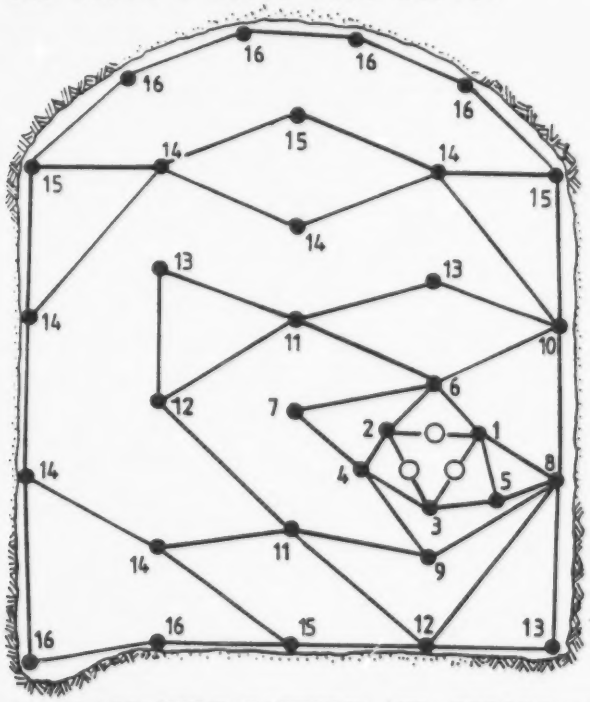


FIGURE 5.7 BLAST HOLE PATTERN AND FIRING SEQUENCE

Several mines in Australia (Enright, 1987) and Canada (McKenzie, 1988) have reported an increase in dust explosions when two adjacent or nearby drives were blasted. The ground vibrations or air-blast caused by blasting in one drive may disturb previously deposited dust in the second drive and lead to the formation of an explosive dust cloud prior to blasting in this drive.

DUST EXPLOSIONS IN RAISES

Detonation of explosives in the blast holes of a raise can lead, in a manner similar to that described in a drive, to the generation and ignition of a dust cloud. In addition, dust clouds may be generated by one or more of the following methods:

1. As the broken rock falls down the raise it may disturb previously deposited dust on the walls of the raise.
2. Broken rock will generate a dust cloud as it strikes the bottom of the raise.
3. Hot explosion gases venting through the holes of a long-hole raise may disturb dust in the drive above the raise.

The dust cloud generated in a raise will take longer to settle out than a dust cloud in a drive. Ignition sources may be active in a raise for a considerable length of time such as when burning particles fall down the raise.

At least one, and possibly more dust explosions have occurred in drives above long-hole raises. The explosion appears to have been caused by hot gases venting through the long holes and disturbing previously deposited dust in the drive. A contributing factor may have been the use of drill cuttings as stemming in the blast holes. There appeared to be no evidence of a dust explosion in the lower drive where the blasted rock was deposited. (Byberg, 1988).

5.9 DUST EXPLOSIONS ASSOCIATED WITH SECONDARY BLASTING

Oversized boulders at drawpoints, ore passes and crushers, are usually broken by placing a charge on the surface of the boulder and detonating it or by drilling a hole in the boulder which is then charged and blasted. When rock is broken by either method the air blast caused by detonation of the explosive may disturb nearby dust and ignition of this dust may cause a dust explosion. A dust cloud can also be produced when detonating cord is used to fire one or more secondary blasts and this cloud can be ignited when the unconfined explosive is detonated. The primary dust explosion can lead to a violent secondary dust explosion, especially if the blasted rock is near a comparatively dry and dusty drawpoint or stope. The use of a surface charge appears to be more conducive to the development of a dust explosion than drilling and blasting as around four times more explosive is used in plastering and confinement of the explosive is minimal.

Hangups at drawpoints are often freed by detonating several sticks of explosive. The explosive is usually tied to a wooden or plastic rod and forced as far as possible up the drawpoint. It is difficult to wash down rock located some distance up a drawpoint and detonation of the unconfined explosive may result in a primary dust explosion in the drawpoint and a secondary dust explosion in the stope.

In many mines secondary blasting of oversized boulders and the use of explosives to free drawpoints are permitted during a normal working shift when miners are underground. The dangers associated with these practices should be recognised especially when a potential primary dust explosion has access to a mass of fine dust as would be found in a dry partially excavated stope.

5.10 GASEOUS EXPLOSION PRODUCTS

HAZARDS ASSOCIATED WITH SULPHUR DIOXIDE

The toxic effects of sulphur dioxide have been responsible for the majority of fatalities and serious injuries caused by sulphide dust explosions. The magnitude of the problems associated with sulphur dioxide are shown in the review of the dust explosions described in Chapter 3. For example, the volume of sulphur dioxide produced during the Fox mine explosion was such that the surface mill had to be evacuated when the gas exhausted from the mine. Significant concentrations of hydrogen sulphide following sulphide dust explosions have been reported at mines in the United States (Gardner et al., 1926) and Canada (McKenzie, 1988).

A review of the physical and toxic properties of sulphur dioxide, hydrogen sulphide and carbon monoxide is given below. Carbon monoxide has been included as this gas has been responsible for the majority of deaths associated with coal dust explosions. The introduction of a filter-type self-rescuer designed to convert carbon monoxide to carbon dioxide has been credited with a dramatic reduction in the number of deaths following coal dust explosions.

PHYSICAL PROPERTIES OF SULPHUR DIOXIDE, HYDROGEN SULPHIDE AND CARBON MONOXIDE

Sulphur dioxide and hydrogen sulphide provide warnings of toxic concentrations due to their pungent or offensive odour. Carbon monoxide is a colourless, odourless gas which provides little or no indications of toxic concentrations. The physical properties of the gases are given in Table 5.7.

Table 5.7: Physical Properties of SO_2, H_2S and CO

	SO_2	H_2S	CO
Mol. Wgt.	64.06	34.09	28.01
Specific Gravity[1]	2.264	1.189	0.968
Colour	colourless	colourless	colourless
Odour	pungent, suffocating	offensive	odourless
Solubility in 100 parts	22.8 g @ 0°C	437 cc @ 0°C	3.5cc @ 0°C
e [2]	non-flammable	4.3%	12.5%
ue [3]	non-flammable	46%	74.2%
Autoignit. temp.		260°C	1204°C

NOTES:

1. Relative density with reference to air = 1.0
2. Lower explosive limit
3. Upper explosive limit

TOXICOLOGY OF SULPHUR DIOXIDE, HYDROGEN SULPHIDE AND CARBON MONOXIDE

The toxic effects of a gas are related to a number of factors including time exposed, gas concentration and work being performed. An individual will breathe around 5 litres/min at rest and 20 litres/min with heavy exertion. High temperatures, humidity, exercise and emotional stress will tend to increase the breathing and heart rate.

Toxicity is the ability of a chemical molecule or compound to produce injury once it reaches a susceptible site in or on a body. Toxicity hazard is the probability that injury may be caused by the manner in which the substance is used. The toxic hazard rating of sulphur dioxide, hydrogen sulphide and carbon monoxide based on data published by Sax et al. (1963) are given in Table 5.8 . The following definitions apply to terms used in Table 5.8.

Acute - This term is used in the medical sense to mean "of short duration". It refers to a single exposure of a duration measured in seconds, minutes or hours when substances are inhaled or absorbed through the skin.

Chronic - This terms is used in contrast to "acute" and means "of short duration". It refers to prolonged or repeated exposures of a duration measured in days, months or years. The term "chronic" does not refer to severity of symptoms but carries the implication of exposures or doses which would be relatively harmless unless extended or repeated over long periods of time.

Local - This term refers to the site of action of an agent and means that the action takes place at the point or area of contact. The site may be skin, mucous membranes of the eyes, nose, mouth, throat or anywhere along the respiratory or gastrointestinal system. Absorption does not necessarily occur.

Systemic - This term refers to a site of action other than the point of contact and presupposes that absorption has taken place. A material is said to have been absorbed only when it has gained entry into the blood stream and consequently may be carried to all parts of the body.

Table 5.8: Toxic Hazard Rating of SO_2, H_2S and CO

	SO_2	H_2S	CO
Acute Local	irritant 3 inhalation 3 ingestion 3	irritant 3 inhalation 3	0
Acute Systemic	U	inhalation 3	inhalation 3
Chronic local	irritant 2 inhalation 2	irritant 3	0
Chronic systemic	U	inhalation 3	inhalation 3

Toxic Hazard Rating Code

0 None: (a) No harm under any conditions;
(b) Harmful only under unusual conditions.

1 Slight: Causes readily reversible changes which disappear after end of exposure

2 Moderate: May involve both irreversible and reversible changes, not severe enough to cause death or permanent injury.

3 High: May cause death or permanent injury after very short exposure to small quantities.

U Unknown: No information on humans considered valid by authors. (Sax et al. 1963).

Sulphur dioxide

The gas can be detected by the average individual at 3ppm and concentrations of 6 to 12ppm cause immediate irritation to the nose and throat. The gas is dangerous to the eyes as it causes irritation and inflamation of the conjunctiva. A concentration of 20ppm is the least amount which is irritating to the eyes. The gas has a suffocating odour, and is a corrosive and poisonous substance. In moist air or fogs it combines with water to form sulphurous acid, but is only very slowly oxidized to sulphuric acid.

Sulphur dioxide affects the upper respiratory tract and the bronchi. It may cause edema of the lungs or glottis, and can produce respiratory paralysis. The gas is so irritating that it provides its own warning of toxic concentrations. Concentrations of 0.1 to 1ppm can be detected by the average individual, possibly by taste rather than the sense of smell. A concentration of 400 to 500ppm is immediately dangerous to life and 50 to 100ppm is considered to be the maximum permissible concentration for exposures of 30 to 60 minutes. Less than fatal concentrations can be borne for fair periods of time with no apparent permanent damage. The toxicity of sulphur dioxide is comparable to that of hydrogen chloride.

Hydrogen sulphide

Hydrogen sulphide is both an irritant and an asphyxiant. Low concentrations from 20 to 150ppm cause irritation of the eyes; slightly higher concentrations may cause irritation to the upper respiratory tract, and if exposure is prolonged, pulmonary edema may result. With higher concentrations the action of the gas on the nervous system becomes more prominent, and a 30 minute exposure to 500ppm results in headache, dizziness, excitement, staggering gait and dysuria, followed sometimes by bronchitis or bronchopneumonia. The action on the nervous system is, with small amounts, one of depression; in larger amounts, it stimulates, and with very high amounts of 800 to 1000ppm may be fatal in 30 minutes, and higher concentrations are instantly fatal. Fatal hydrogen sulphide poisoning may occur even more rapidly than that following exposure to a similar concentration of hydrogen cyanide. Hydrogen sulphide does not combine with the hemoglobin of the blood; its asphyxiant action is due to paralysis of the respiratory centre.

Carbon Monoxide

Carbon monoxide has an affinity for hemoglobin 210 times that of oxygen, and by combining with the hemoglobin, renders the latter incapable of carrying oxygen to the tissues. The effect on the body is therefore predominantly one of asphyxia. In addition to this action, the presence of CO - hemoglobin in the blood interferes with the dissociation of the remaining oxy-hemoglobin, so that tissues are further deprived of oxygen. A concentration of 400 to 500ppm in the air can be inhaled without appreciable effect for 1 hour. A concentration of 1,000 to 2,000ppm is dangerous and concentrations of 4,000 and over are fatal in less than an hour. Carbon monoxide is considered to be particularly dangerous as it is almost impossible to detect by smell or taste.

6. INHIBITION OF COAL DUST EXPLOSIONS

6.1 INTRODUCTION

Techniques applied in the mining industry to prevent sulphide dust explosions include the use of limestone to inert sulphide dust, water sprays and foggers to increase the moisture content of the dust, and limitations on the overall blasting time with conventional mining explosives. These techniques were based on those used in coal mines to prevent coal dust explosions. Several modifications were introduced in adapting the preventive measures and it is possible that some of the original criteria considered to be essential to prevent coal dust explosions have been neglected. It is important to review the methods used in coal mines to prevent dust explosions and consider the suitability and limitations of these methods in the mining of sulphide ore bodies.

6.2 COAL DUST EXPLOSIONS

In a modern mine coal is won from the face using continuous miners in room and pillar workings and shearers in longwall faces. Cutting and shearing of the coal produces dust continuously during the shift and airborne dust is transported by the ventilating air and settles on the underground workings away from the face. This coal dust can form an explosive mixture when raised into the air. The mass of coal dust needed to propagate an explosion is small, around 200 g/m^3 and a dust layer around the thickness of a piece of paper over the surfaces of a 2m x 5m roadway would provide sufficient dust for an explosion.

When coal is drilled and blasted from a seam a highly combustible dust cloud is produced, and restrictions are placed on the types of explosives, number of holes and the delay between shots. The explosives used, termed "permitted

explosives", have a flame depressant additive which reduces the volume, duration and temperature of the explosion flame. The interval between firing the first and last charged hole is usually restricted to less than 200 ms.

In order to discuss any aspect of a coal dust explosion, whether it be the development of pressure and flame in a gallery or the action of suppressive devices, it is necessary to establish a model of the events taking place and of the processes going on during an explosion.

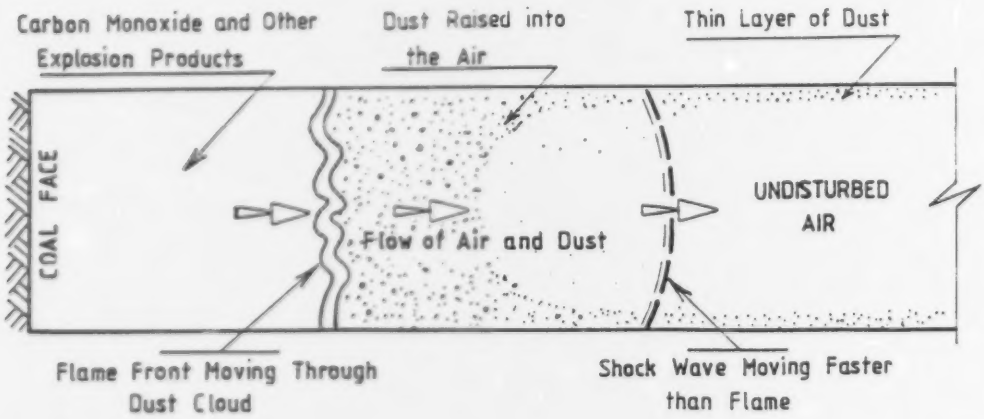

Carbon Monoxide and Other Explosion Products
Dust Raised into the Air
Thin Layer of Dust
COAL FACE
Flow of Air and Dust
UNDISTURBED AIR
Flame Front Moving Through Dust Cloud
Shock Wave Moving Faster than Flame

FIGURE 6.1 COAL DUST EXPLOSION

Figure 6.1 shows an idealized picture of a fully developed self-sustaining coal dust explosion in a mine roadway. To the right of the shock wave the air and dust in the roadway are undisturbed and the air velocity is zero. At the shock wave the air velocity is raised instantaneously and remains constant between the shock front and the flame front. The shock wave is commonly referred to as the pioneer wave and it is this wave which violently disturbs the dust throwing it into suspension in the roadway.

The static pressure ahead of the shock wave is the atmospheric pressure. At the shock wave the pressure rises and then remains constant until the flame front arrives and the pressure falls. At some distance behind the shock wave the flame front reacts with the coal dust-air mixture. This flame front travels at a velocity less than the shock front but greater than the unreacted air-dust velocity and combustion of the coal dust provides the energy necessary for the continuous propagation of the explosion. There are two schools of thought on the actual combustion of the coal dust. The older theory assumes that the volatile components of the dust are distilled off and the resulting gases then burn. The other theory suggests that surface oxidation takes place and the rate of reaction is controlled by diffusion of oxygen to the surface particles. (Hartmann, 1957).

In general, two types of explosions are possible; these are termed deflagrations and detonations. The basic difference between these two reactions is in their mode of propagation. In a deflagration the reactants are heated by heat transfer from the flame and the flame velocity and dust-air mixture velocity may vary between 50 m/s to 1,100 m/s. With a detonation the reactants are heated by the action of the shock wave and immediately behind the detonation the products move with the velocity of sound relative to the detonation. For a coal dust detonation, the shock front and the flame front are close together, travel at speeds around 2,400 m/s and develop pressures in excess of 27 atmospheres.

The usual sequence in the development of a coal dust explosion is when a local ignition of a methane-air mixture causes a gas explosion and the shock wave from this explosion disturbs dust previously deposited on the roof, ribs and floor of the workings. The flame front associated with the gas explosion ignites the airborne dust and a

secondary, and much more violent, self-sustaining dust explosion is produced. When explosives are used to blast the coal from the face, then the flame associated with the explosives may ignite the dust cloud produced during blasting. This comparatively minor explosion may lead, in the manner described for a methane-air explosion, to a self-sustaining secondary explosion.

Defensive measures used in coal mines to prevent dust explosions include restrictions on the type and use of explosives, elimination of ignition sources, ventilation to remove methane, the application of inert dusts or water to reduce the explosibility of a coal dust. A second line of defense is the use of passive and active barriers to arrest a developed explosion by quenching the flame front with inert dust or water behind the shock wave.

The final measure used to protect miners from the toxic gases produced during a dust explosion is the issue to miners of portable self-contained self-rescuers designed to convert carbon monoxide to carbon dioxide.

6.3 EFFECT OF ADDITION OF INERT DUSTS

The mixing of an inert dust with coal dust causes a decrease in the maximum explosion pressure and rate of pressure rise and a non-explosive mixture will be obtained when sufficient inert dust is mixed with the coal dust. The results of a series of tests in an 8 l. explosion chamber with mixtures of coal and limestone dust are shown in Figure 6.2 (Enright, 1984). These results are somewhat disturbing as they indicate that the addition of 35 percent limestone, intimately mixed with the coal, has little effect on reducing the explosion pressure and high explosion pressures, in excess of 500 kPa, can be obtained with a mixture of 70 percent coal and 30 percent limestone. Similar results have been reported by other investigators. (Cybulski

1975, Richmond et al. 1975, Hertzberg et al. 1982). Results similar to those shown in 6.2 were obtained with coal-sand and coal-salt mixtures.

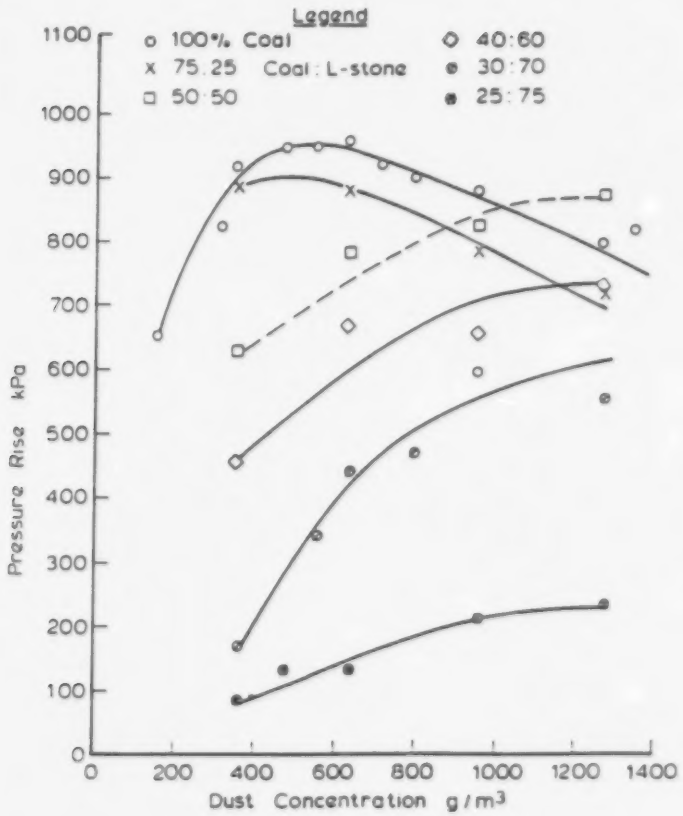


FIGURE 6.2 PRESSURE RISE VERSUS DUST CONCENTRATION COLLIE COAL-LIMESTONE DUSTS (Enright, 1984)

The addition of inert dust inhibits the development of a dust explosion by several different methods:

1. Combustion of the fuel is retarded as portion of the heat of combustion is transferred to the inert dust.
2. Inert dust inhibits the diffusion of oxygen to the burning particles and restricts the diffusion of gases from the particles.

3. The inert particles shield the fuel from radiation energy produced by the reaction of the combustion of other burning particles.
4. Addition of an inert dust decreases the percentage of combustible dust in the mixture.
5. When limestone is used as an inhibitor the decomposition of calcium carbonate occurs as an endothermic reaction causing the absorption of heat during an explosion. The carbon dioxide produced during the reaction may further inhibit the explosion.

6.4 MASS OF LIMESTONE REQUIRED TO INERT COAL DUST

The effects of the addition of limestone to Collie and Newcastle coal dust samples are shown in Figure 6.3. In this figure the maximum pressure developed for a particular dust mixture is plotted against the total incombustible weight percent of the dust. The curves show that greater than 75 percent limestone was required to inert the coals under test. With a mixture of 30 percent Collie coal and 70 percent limestone (77 wt. pct. inert) a maximum explosive pressure of 600 kPa was recorded. Similarly a mixture of 30 percent Newcastle coal and 70 percent limestone (75 wt. pct. inert) developed a maximum pressure of 480 kPa (Enright, 1984). Values reported by Hertzberg et al. (1982) for tests on Pittsburgh coal in an identical 8 l. chamber are also shown in Figure 6.3.

In order to render a coal dust incapable of propagating an explosion a high content, exceeding 80 percent in some cases, of incombustible matter is required. Coal dusts containing high percentages of inert dust, close to the maximum value required to completely inert the coal dust, are still capable of propagating an explosion. However, lower percentages of inert dust are of value in inhibiting coal dust explosions since the violence of the explosion is reduced. The increase in the lower limit explosive concen-

tration is of practical value as the shock wave to produce a dust concentration of 500 g/m³ is greater than that required to produce a concentration of 200 g/m³. Thus a small methane-air explosion may not produce sufficient energy to create a dust cloud within the explosive concentration limits.

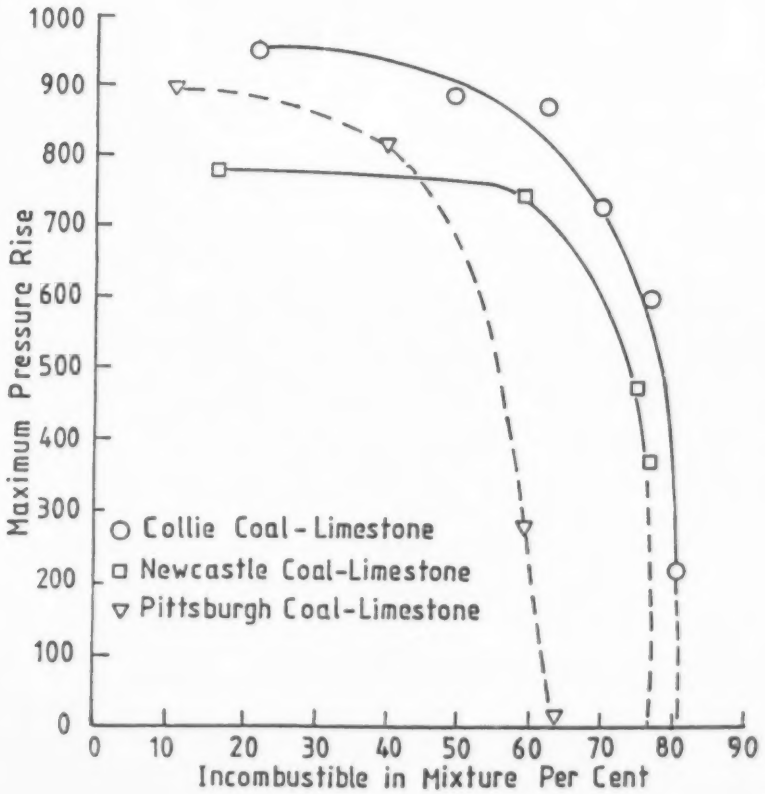


FIGURE 6.3 EFFECT OF ADDED LIMESTONE ON PRESSURE RISE. (Enright, 1984)

The New South Wales Mines Regulation Act (N.S.W. Government 1982) states that sufficient inert dust must be applied so as to ensure that the resulting mixture of coal and inert dust contains at least 75 percent incombustible

matter at the face workings and 65 percent throughout the remainder of the mine. This percentage includes the ash forming constituents and inherent moisture of the coal.

6.5 SIZE ANALYSES OF LIMESTONE

The use of limestone to inhibit a coal dust explosion is only effective when the limestone is easily dispersed by an air blast and remains in suspension for some time. The N.S.W. Mines Regulation Act (1982) states that crushed limestone used to inhibit dust explosions shall be:

(i) Of such fineness that, of the dry dust which passes through a 250 micrometre aperture not less than 50 percentage by mass and, except with the permission in writing of the Chief Inspector, not more than 75 percentage by mass shall pass through a 63 micrometre aperture;

(ii) Of such character that it is readily dispersable into the air and, when in use in places where it is not directly wetted by water from the strata, does not cake but is dispersed into the air when blown upon with the mouth or by a suitable appliance.

A particle size analysis of the limestone dust used in N.S.W. is given in Table 6.1.

Table 6.1: Size Analysis of Limestone Dust

Sieve Aperture micrometres			Percent retained on sieve
	+	125	6.9
-125	+	90	15.3
- 90	+	45	41.5
	-	45	36.3

6.6 USE OF WATER TO INHIBIT COAL DUST EXPLOSIONS

The use of water sprays to reduce the mass of respirable dust and inhibit the development of coal dust explosions is a common practice in coal mines. The method by which free water, as distinct from inherent moisture, acts in reducing the explosibility of a coal dust is somewhat different from that of stonedust. Free water absorbs heat from the flame due to its high specific heat, 0.24 kJ/kg, and heat of vaporization, 128.3 kJ/kg, whereas limestone with a lower specific heat, 0.05 kJ/kg, tends to shield coal particles from radiant energy and the reaction producing calcium oxide and carbon dioxide requires energy.

An additional effect of water is that cohesion increases with moisture and agglomeration of coal dust particles occurs. Various researchers (Bartknecht, 1978, Nagy and Verakis, 1983) have shown that the explosion parameters of maximum pressure developed, Pmax, and rate of pressure rise, (dp/dt)max, decrease with an increase in particle size. The increased moisture content reduces the dispersibility of the dust and at high moisture contents a mud like mixture may be formed. The ability of a coal to absorb moisture varies between samples. Coals with a high inherent moisture may absorb water more readily than those with low inherent moistures.

Another effect of moisture on dust explosibility is that an increase in free water increases the minimum ignition energy of a dust cloud and very strong initiator sources may be required at high moisture contents.

The addition of moisture to coal dust reduces the maximum pressure and rate of pressure rise in a similar manner to that caused by the admixture of an inert dust. (Hertzberg et al. 1979, Enright, 1985). The superimposed pressure-time curves shown in Figure 6.4 were obtained with

Wyvern coal dust in an 81 explosion chamber. High explosion pressures were observed with coal containing 39 percent water but beyond this value the mixture formed a slurry and could not be dispersed with the experimental equipment. Cybulski (1975) found that coal containing water in excess of its absorbing power propagated an explosion of tremendous violence when the initiator was sufficiently strong. Eckhoff and Mathisen (1978) reported that tests by Scholl in a $1m^3$ vessel revealed that peat dust with a moisture content around 50 percent was capable of producing an explosion.

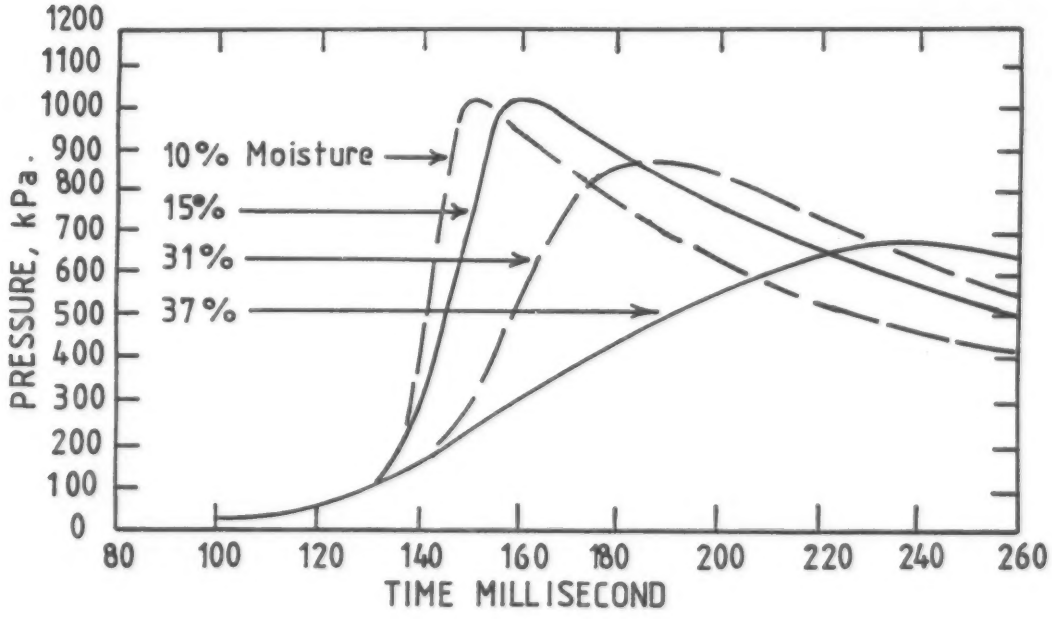


FIGURE 6.4 PRESSURE - TIME CURVES AT SELECTED MOISTURE VALUES - WYVERN COAL

6.7 PERMITTED EXPLOSIVES IN COAL MINING

Explosives designated "permissible" are those which have been tested and approved for use in underground coal mines. The explosives have a flame depressant additive, such as sodium chloride, which reduces the volume, duration and temperature of the explosion's flame. They are designed

to minimize the possibility of a gas or dust ignition in underground coal mines.

The number of shots in a coal face at any one time is usually restricted by Government regulations. In New South Wales the maximum number of shots is six unless special permission is given by the Chief Inspector. Instantaneous detonators must be used in all shots.

6.8 STONEDUST AND WATER BARRIERS

Limestone is applied to the surfaces of a coal mine in sufficient quantities so that in the event of an explosion the mixture of limestone and coal thrown into the air cannot be ignited by the flame front of the explosion.

Stonedust and water barriers are designed to arrest an explosion by quenching the flame travelling behind the shock wave of an explosion. Stonedust barriers consist of lightweight wooden shelves placed loosely across a roadway and the shelves are heaped with large quantities of stonedust. The shelves are designed so that the velocity and pressure of the air immediately following the shock wave disturb the shelves and a dense cloud of stonedust fills the roadway prior to arrival of the flame front.

The stonedust barrier shown in Figure 6.5 is typical of the design used in coal mines. The barrier consists of 10-20 sets of loaded shelves laid across a 5m wide by 3m high roadway. The loading on each set of shelves is 270 kg of limestone and the distance between each set is 3m.

Two conditions must be satisfied for the successful operation of a stonedust barrier:

(a) The air pressure and velocity immediately behind the shock wave must be sufficient to disturb the barrier. Thus the barrier must be designed so that it will respond to a minimum increase in air pressure or velocity but still remain stable during normal working conditions.

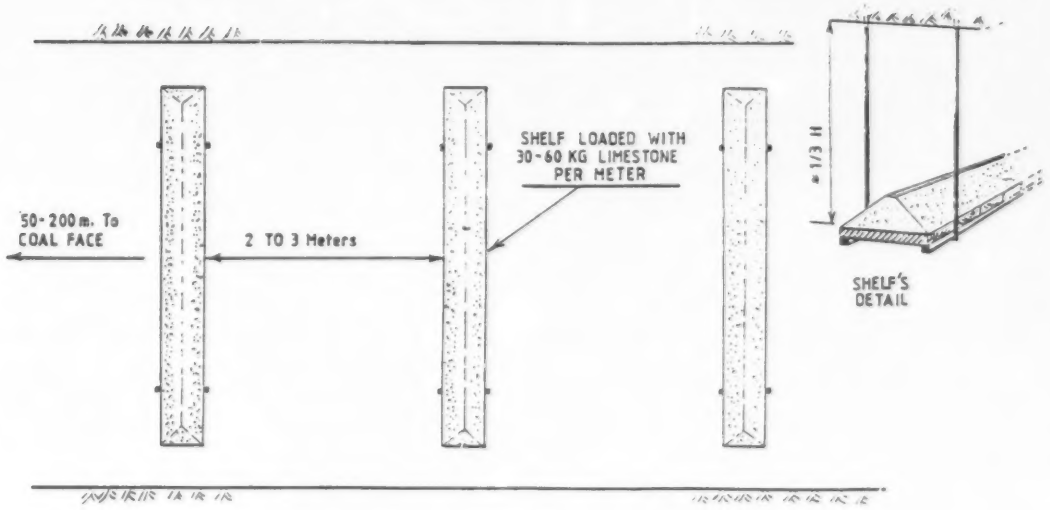


FIGURE 6.5 PART OF BARRIER SHOWING 3 SHELVES HUNG ACROSS ROADWAY (TOTAL NUMBER OF SHELVES 10 TO 20)

(b) The flame front must arrive at the barrier while stone dust is in suspension. This requirement is difficult to fulfill as the air velocity behind the shock front and the flame velocity vary during the life of an explosion. As a consequence the interval between arrival of the shock front and the flame front varies with their respective velocities. During a self-sustaining explosion in a mine the shock wave velocity and the flame velocity increase as the explosion travels along a roadway. The shock wave velocity may increase from 350 m/s to excess of 1000

m/s and the flame velocity from around 10 m/s to greater than 600 m/s.

The following examples illustrate the problems associated with the design of stonedust barriers.

Case 1 - Slow Explosion

A dust explosion develops at the end of a mine roadway with the following properties:

Shock wave velocity = 375 m/s

Flame velocity = 50 m/s

The relative arrival times of the shock wave and flame front along the mine roadway are given in Figure 6.6. If a stone dust barrier was located 200m from the end of the roadway, then the interval between the arrival of the shock wave and the flame front would be 3.47 seconds. The conditions in the roadway when the shock wave has reached this position are shown in Figure 6.7.

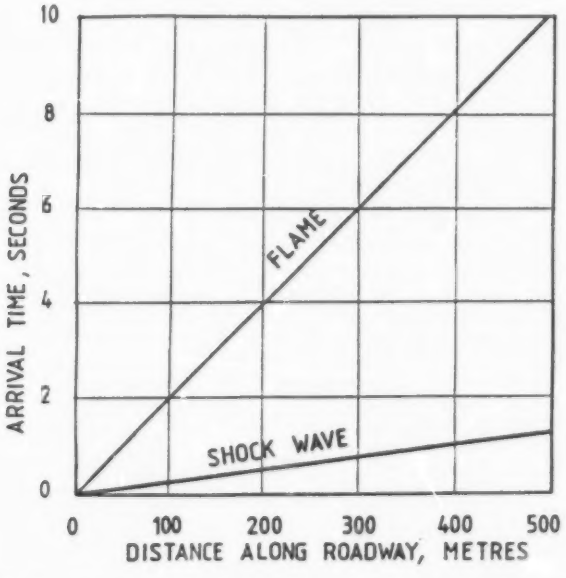


FIGURE 6.6 CASE I - ARRIVAL TIMES OF SHOCK WAVE AND FLAME FRONT.

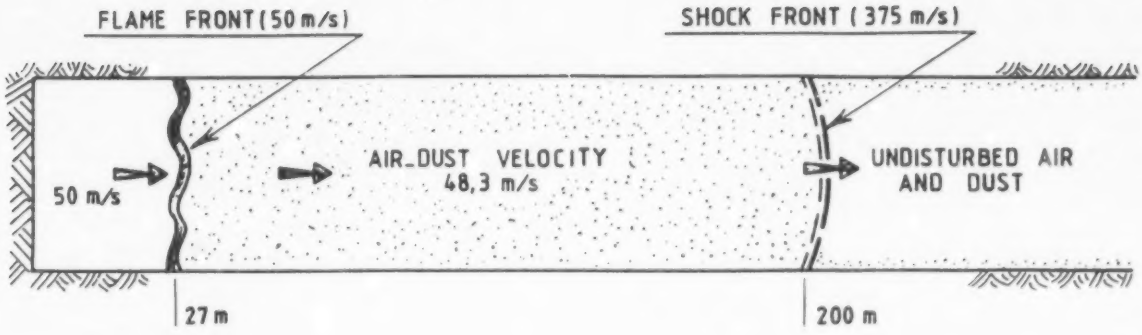


FIGURE 6.7 CASE I - CONDITIONS WITH SHOCK WAVE AT 200m

Case II - Fast Explosion

Shock wave velocity = 610 m/s

Flame velocity = 400 m/s

The relative arrival times of the shock wave and the flame front along the mine roadway are shown in Figure 6.8. The time interval between arrival of the shock wave and flame front for a barrier located 200m from the end of the roadway would be 0.17 seconds. Figure 6.9 shows the conditions in the roadway when the shock wave reaches the barrier.

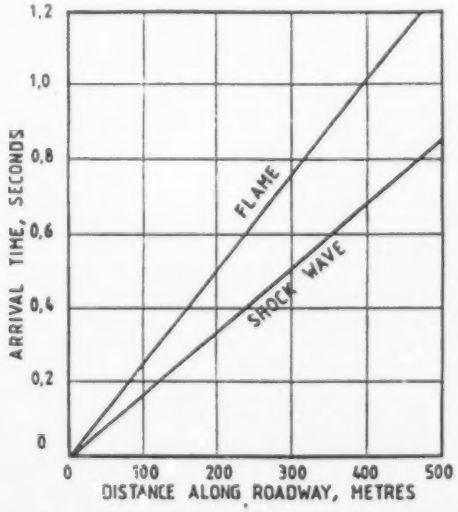


FIGURE 6.8 CASE II - ARRIVAL TIMES OF SHOCK WAVE AND FLAME FRONT

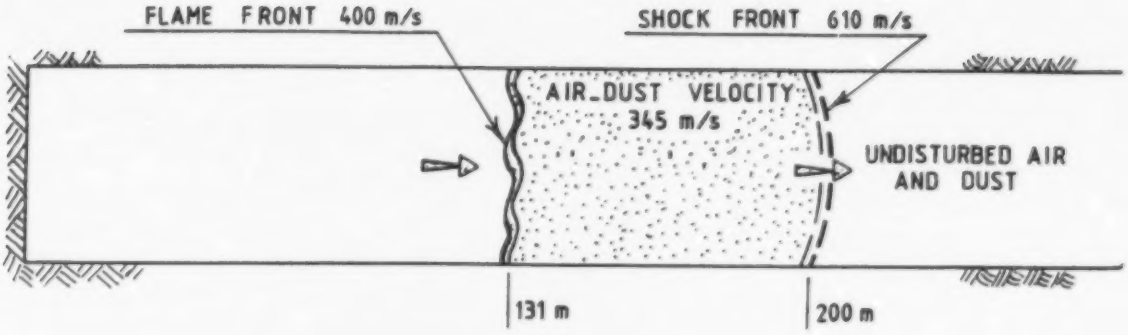


FIGURE 6.9 CASE II CONDITIONS WITH SHOCK FRONT AT 200m

A comparison between the slow and fast explosions shows that, for a barrier located at 200m, the interval between the shock wave and flame front is reduced from 3.47 seconds to 0.17 seconds. In Case I the barrier would be ineffective as the dust cloud would tend to settle out during the interval between the shock front and the flame front. In Case II the flame front would pass the barrier before it had time to operate.

The importance of the location of the barrier with respect to the point of origin of the explosion can be shown with Case I. The time interval between the arrival of the shock wave and flame front at various distances along the roadway is as follows:

Distance from Face	Time Interval
50 m	0.87 sec
100 m	1.73 sec
150 m	2.60 sec
200 m	3.47 sec
300 m	5.20 sec
400 m	6.93 sec

Government regulations usually specify that the barrier must be installed so that one end of the barrier is within 200m of the coal face and no closer than 60m to the face.

Water barriers are designed to act in a similar manner to stonedust barriers. The individual water troughs are designed to shatter under the effects of the air velocity and pressure following a shock wave.

In New South Wales the total quantity of stonedust or water contained on any type of barrier shall not be less than 400 kg per square metre of roadway cross-sectional area in a Class A seam. (N.S.W. Government, 1984). Thus the total barrier loading in a 5m x 3m heading would vary between 3,000 and 6,000 kg of water or limestone depending on the nature of the seam.

7. PREVENTION AND INHIBITION OF SULPHIDE DUST EXPLOSIONS

7.1 COMPARISON BETWEEN COAL AND HARDROCK MINING METHODS

There are significant differences between underground coal and metalliferous mining methods. Coal mining is essentially a two dimensional process during which coal is produced continuously from a series of headings and cross-cuts. The dimensions of the headings are usually around 5m by 3m and the exposed surfaces are within easy access to the miners. Since production of the coal is a continuous process, the generation of coal dust is continuous and the freshly exposed surfaces become covered with coal dust. These fresh surfaces are sprayed with limestone, usually after every second or third shift. In addition, previously stonedusted areas are regularly inspected to ensure that adequate amounts of stonedust are present.

The bulk of production from a metalliferous mine comes from open stopes where large tonnages are broken at irregular intervals. The stopes are often 50 to 100m in height and several thousand kilograms of high explosive may be used in a single stope blast. Dust is produced during blasting and as the broken rock strikes the walls and floor of the stope.

Development work in a metalliferous mine is somewhat similar to coal mining in as much as the excavations are of limited cross-sectional area and of considerable length. However, the work is cyclic and around 200 kg of explosive per blast are used to break rock from the face.

The essential requirements for a dust explosion are illustrated in Figure 7.1. A break between any two components in the system will prevent the development of a dust explosion. For example, a dust explosion cannot occur in an oxygen deficient atmosphere regardless of the dust concentration and strength of the ignition source.

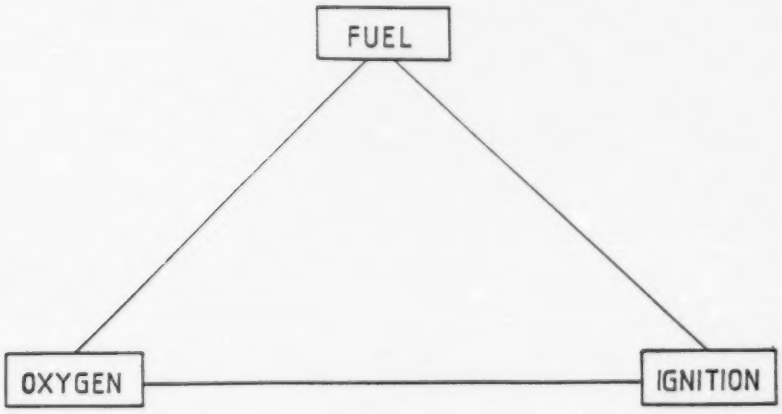


FIGURE 7.1 EXPLOSION TRIANGLE

The following techniques are available to prevent or reduce the effects of a sulphide dust explosion.

1. Remove or inert previously deposited dust which may contribute to a primary or secondary dust explosion.
2. Inert the combustible dust produced during blasting.
3. Quench the flame of an existing dust explosion.
4. Limit the effects of the toxic gases (SO_2 and H_2S) following in the wake of an explosion.
5. Reduce the temperature and volume of the explosion gases generated by detonation of high explosives and thus prevent ignition of the cloud produced during blasting.
6. Reduce the oxygen content in an underground excavation with the inert gases produced during blasting.

The objectives of any system used in a particular mine to prevent dust explosions should be clearly defined so that suitable suppression techniques can be developed. Many mines have developed techniques based on those used in coal or other hard rock mines and then modified them to suit local conditions. The end result, usually third or fourth generation, is often an ineffective and sometimes costly explosion suppression system.

7.2 USE OF LIMESTONE TO INERT SETTLED DUST

Data presented in Section 4.5,(Figure 4.6) and Section 5.3 (Figures 5.2 and 5.3), indicate that it is necessary to reduce the sulphur content of a dust to less than 20 percent in order to render the dust non-explosive. The effective use of limestone requires knowledge of the mass of sulphide dust to be inerted and an efficient method to adequately mix the two dusts. In most applications in hard rock mines the use of limestone begs the question of how much sulphide dust is to be inerted.

Accurate information on the mass of dust lying on the surfaces of underground excavations and the mass produced during blasting is extremely limited. Theoretical calculations indicate that the mass of dust in the excavations may be several times larger than that previously considered by many mining companies. Consider an orebody with the following properties:

Sulphide content of ore	=	40 percent
Density of ore	=	4500 kg/m^3
Density of dust	=	3000 kg/m^3

The mass of dust per square metre of surface area for various layers is given in Table 7.1. Also included in the table is the mass of dust present on the roof and walls of a

stope void 25m wide, 25m long and 50m high with dust layers of various thicknesses (Figure 7.2).

Table 7.1: Mass of Dust on Roof and Walls of a 25 x 25 x 50m Stope Void

THICKNESS OF LAYER	MASS OF DUST PER M²	MASS OF DUST ON ROOF	MASS OF DUST ON WALLS
0.1 mm	0.3 kg/m²	187 kg	1800 kg
0.25 mm	0.75 kg/m²	469 kg	4500 kg
0.50 mm	0.5 kg/m²	938 kg	9000 kg
1.0 mm	3.0 kg/m²	1870 kg	18000 kg

Ore with a 40 percent sulphur content requires one kilogram of limestone per kilogram of dust in order to reduce the sulphur content to 20 percent. It is obvious that, even with a layer of dust less than 0.1mm on the roof, the practice of dispersing two or three 25 kg bags of limestone in an attempt to inert dust on the surfaces of the void cannot be very effective. Not only is the mass of limestone inadequate but it is unlikely that limestone dispersed in front of the face will mix with dust disturbed from the roof and walls.

The mass of dust on the roof and walls in a 5m wide by 3m high drive through the orebody would vary from 3.3 kg per metre length for a layer 0.1mm in thickness to 33 kg for a 1mm layer. From a practical point of view an adequate mass of dust could be mixed with the sulphide dust to lower the sulphur content to less than 20 percent. A simpler, cheaper and more effective suppression technique would be to wash the dust from the roof and walls.

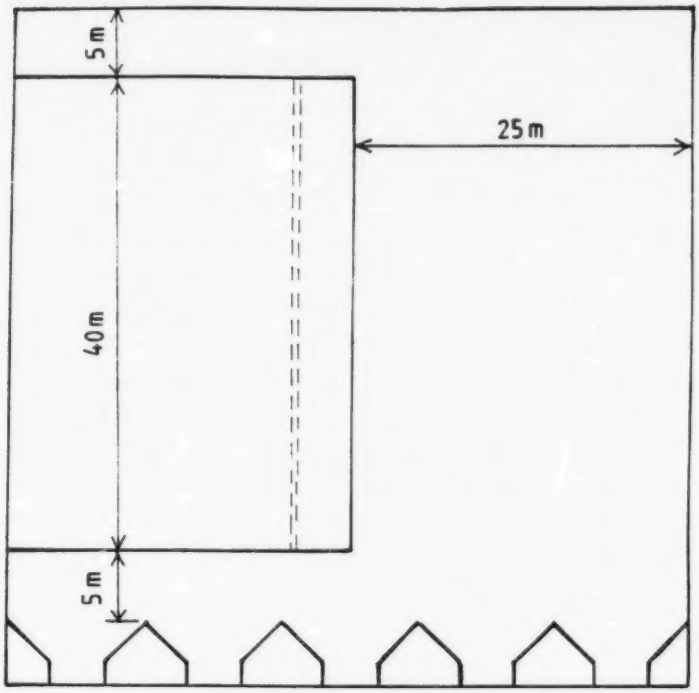


FIGURE 7.2 SECTION THROUGH STOPE

7.3 USE OF LIMESTONE TO INERT DUST GENERATED DURING STOPE BLASTING

The model of a stope blast discussed in Section 5.7 indicated that the following separate mechanisms are involved in the generation of dust clouds during and after detonation of an explosive column.

* Gases venting through the top or bottom of a blast hole may carry dust from the pulverized zone of the blast hole.

* A dust cloud is generated when broken rock collides with the walls of a stope.
* A dust cloud is generated when broken rock hits the floor of a stope.
* In-flight collisions of the blasted rock will create dust.

Generation of the dust cloud from the pulverized zone may occur within milliseconds of detonation of the explosives while the collision of broken rock with the floor may continue, depending on the stope geometry, for some seconds after detonation.

It is difficult to estimate the mass of dust produced by rock collision. The following analysis is an attempt to estimate the mass of dust produced in the pulverized zone of a blast hole. As discussed in Section 5.6 little data has been published on the mass of dust produced during blasting. Langefors and Kihlstrom (1967) estimated that with a hole diameter of 40mm the thickness of the crushed zone was of the same order or slightly less than the radius of the hole, while Gustafsson (1973) estimated that the diameter of the pulverized section around a 100mm hole was 150mm. Assumptions used in calculating the data given in Table 7.2 are as follows:

(i) Ore density is 4500 kg/m^3.
(ii) Sulphur content of the ore is 40 percent.
(iii) Thickness of the pulverized zone is 25mm regardless of the blast hole diameter.
(iv) Twenty-five percent of the pulverized rock from the crushed zone is less than 400 microns in size and is discharged unaltered in chemical composition into the space around the blast hole.
(v) Density of limestone dust is 1660 kg/m^3.

The calculated mass of combustible dust generated by a column of explosive one metre in length is given in Table 7.2. The column height of limestone dust required to inert the combustible dust produced by a one metre column of explosive is also given in Table 7.2.

Table 7.2: Combustible Dust per m of Blast Hole and Column Height of Limestone to Inert Dust

HOLE DIAMETER	CALCULATED MASS OF COMBUSTIBLE DUST kg/m LENGTH OF HOLE	COLUMN HEIGHT OF LIMESTONE TO INERT COMBUSTIBLE DUST
50 mm	6.6 kg/m	2.02 m
100 mm	11.1 kg/m	0.88 m
150 mm	15.5 kg/m	0.53 m
200 mm	19.9 kg/m	0.38 m

The theoretical calculations summarized in Table 7.2 indicate that a considerable mass of fine dust may be produced in the crushed zone of a blast hole. If this dust is discharged into the space above and below the hole, as when inadequate stemming is used, then ignition of the dust by the venting of hot explosion gases would cause a dust explosion. A considerable quantity of limestone is required to inert the theoretical mass of discharged dust. For example, a 150mm diameter hole would require 15 kg of limestone with a column rise of 0.53m for each metre of charged hole.

Based on the preceding assumptions it seems that two or three metres of limestone dust placed at the collar and toe of a blast hole will not be an effective method of preventing a dust explosion. Limestone used as a deck spacing in a blast hole may be of limited value in preventing dust explosions. The bulk of this limestone will

be mixed with the inert explosion gases in the void behind the rock mass and a dust explosion cannot occur in the void until air replaces the inert gases.

7.4 USE OF LIMESTONE IN DEVELOPMENT HEADINGS

The cut holes in a development face are designed to pulverize and eject rock from a face. In a normal development heading using long delay detonators the elapsed time between firing the first and last cut holes will be around one second and the overall blasting time around six seconds. The blast model of a development heading discussed in Section 6.5 indicated that a dust explosion was most likely to occur when the cut and easer holes fired. The probability of a dust explosion decreased as later holes were fired due to a decrease in oxygen content of the atmosphere in the vicinity of the face.

The mass of pulverized rock ejected from a cut can be estimated from the geometry of the cut holes. A burn cut designed to produce a void with a cross-section of 160mm x 160mm using seven drill holes 3m in length and 50mm in diameter would eject around 160 kg of rock with a relative density of 4.5. If the sulphur content of the rock is 40 percent and half the ejected rock is less than 400 micrometres in size then 80 kg of limestone would be required to inert the dust.

It appears practicable to use limestone to prevent a dust explosion when the cut and easers are fired in a development heading. The mass of limestone dust required to inert the sulphide dust is within reasonable limits but to be effective the limestone must be dispersed in an efficient manner so that it mixes with the ejecting sulphide particles. The main variables are the mass of limestone

required to inert the combustible dust, position of the limestone in the drive, dispersion system and timing of the dispersion system.

Three techniques are currently used in hard rock mining operations to create limestone clouds in development headings. (Enright, 1988).

(i) Detonating cord or a small amount of explosive is used to disperse one or two 20 kg bags of limestone suspended at the face. This method has the advantage that the bags can be placed in front of the cut holes and the venting hot gases and rock particles will tend to mix with and disperse the dust.

(ii) Several bags of limestone are stacked near the face and one or more sticks of explosive are used to rupture the bags and disperse the dust (Figure 7.3).

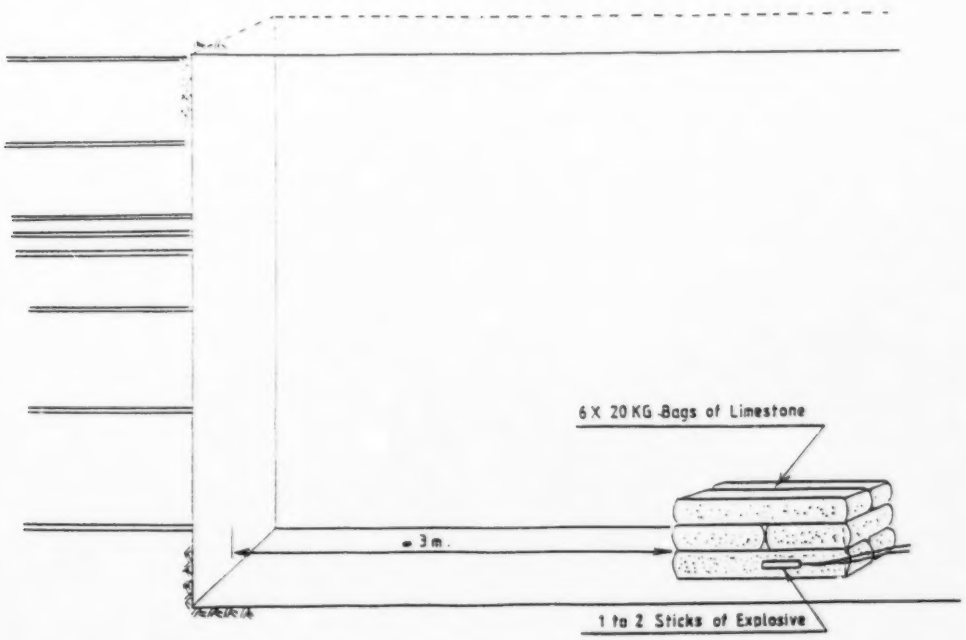


FIGURE 7.3 STACKED BAGS OF LIMESTONE

This technique has found favour with several mines as the bags can be easily and quickly positioned after the face has been charged. However field tests have indicated that with the current techniques few of the bags are ruptured and little dust dispersed. A much more efficient dispersion system is required and tests should also be carried out to determine the optimum distance of the bags from the face.

(iii) Detonating cord is used to disperse a trail of limestone laid on plastic sheeting along a drive (Figure 7.4). Field tests with this method indicated an inefficient dust cloud was generated with three lengths of detonating cord and the presence of water in the drive tended to cause the limestone to cake.

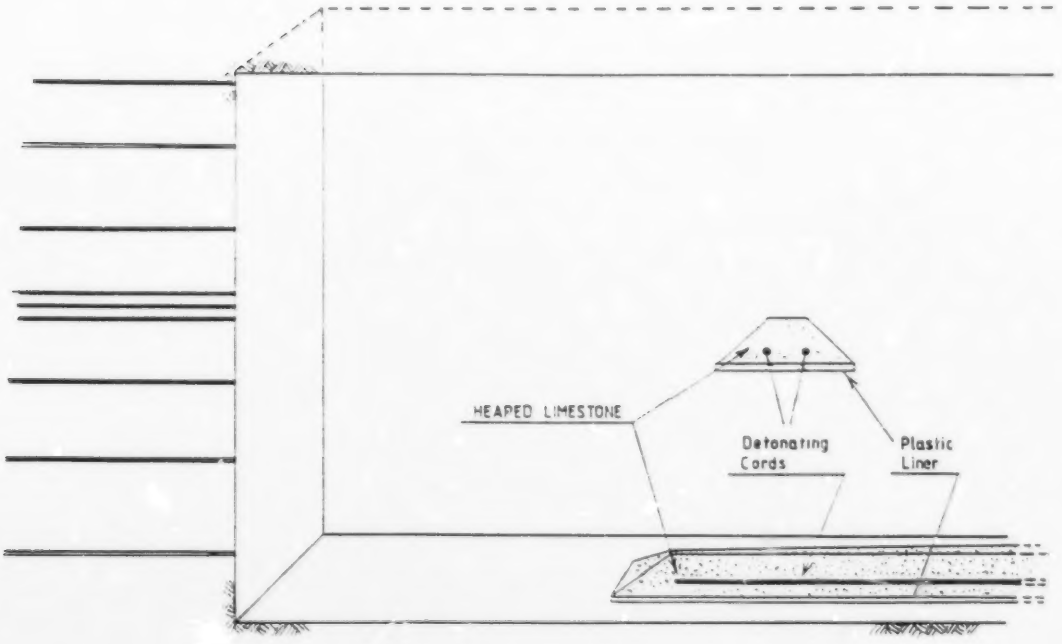


FIGURE 7.4 HEAPED LIMESTONE OVER DETONATING CORDS AND PLASTIC LINER

A fundamental consideration in the design of an efficient dust dispersion system is the need to assess the purpose behind the system. Bags of limestone dispersed next to a face may reduce the temperature of the gaseous explosion products and inert the combustible dust generated during blasting. An inert dust cloud created some distance from the face and along the drive may tend to quench the flame front of a primary dust explosion rather than prevent the development of such an explosion.

7.5 USE OF WATER SPRAYS TO INHIBIT EXPLOSIONS AND DISSOLVE SULPHUR DIOXIDE

Water sprayed on the walls and floor of a stope reduces the explosibility of the previously deposited dust and, due to agglomeration of the wet particles, tends to reduce the mass of dust raised when broken rock hits the walls and floor of the stope. The presence of a dense cloud of micro-droplets of water within a stope may quench the flame of a potential ignition source or the flame front of a developing explosion. Free water is more effective than limestone in quenching the flame front due to its high specific heat, 0.24 kJ/kg, and heat of vaporization, 128.3 kJ/kg compared with the specific heat of limestone, 0.05 kJ/kg.

Water sprays are often installed in development headings, drill drives and drawpoints. The sprays vary in design from flattened lengths of water pipe to commercial foggers using high pressure air and water. It is unlikely that a single spray will produce a sufficient mass of airborne water droplets to quench the flame front of a secondary explosion and a series of sprays may be necessary to achieve this result.

A special application of the sprays may be to dissolve the sulphur dioxide following in the wake of an explosion. A number of strategically placed sprays may isolate a section of a mine contaminated by sulphur dioxide and enable normal operations to continue in the rest of the mine.

7.6 WASHING EXPOSED SURFACES

The simplest and most effective way to prevent a secondary dust explosion in a hard rock mine is to wash settled dust from the surfaces of the underground excavations. This process will not prevent a primary dust explosion caused by ignition of the dust cloud generated during blasting but will certainly ensure that no fuel is available for a secondary explosion.

The surfaces of cut-and-fill stopes and development workings can be cleaned easily with a water jet using normal underground water pressure. Washing down a large open stope is difficult due to the inaccessibility and size of the roof, walls and floor. A high pressure water jet may be required to reach difficult locations in an open stope and the design of new stopes in high risk orebodies should incorporate access drives at selected points to facilitate washing down the surfaces.

7.7 STONEDUST AND WATER BARRIERS

When a dust explosion occurs in an open stope the flame front may travel from the stope void through the drawpoints and the drives connected to the stope. Stonedust or water barriers, similar to those described in Section 6.8, may quench the flame of the existing explosion and prevent the ignition of dust ahead of the flame front.

In an ideal model of a dust explosion (Figure 5.1) the fuel source is ahead of the flame front which is in turn followed by the products of combustion. In such a situation, barriers would not be necessary if the access drives and drawpoints had been washed clean as no fuel would be available for the combustion process to continue.

7.8 USE OF STEMMING TO REDUCE VENT GAS TEMPERATURE AND PRESSURE

At some stage in a blast the explosion gases reach a free face and begin to exhaust into the atmosphere at a temperature Te. This exhaust temperature, Te, is lower than the explosion temperature, Tx, which is of the order of 2000 to 3500°K. The exhaust temperature is a function of several variables including rock type, explosive composition, blast design and the type and weight of stemming materials.

In a series of computer simulations of a drop-raise crater blast Favreau et al. (1986) calculated that the exhaust temperature would be reduced from a high of 3100°K in an unstemmed 6.5 inch (165mm) diameter hole to 2210°K with 5.6 feet (1.7m) of stemming. In the unstemmed case the value of the exhaust temperature corresponded to the explosion temperature since the leading exhaust gas did little work and so remained at a temperature as high as T_x. A saturation effect was observed in that the addition of more and more stemming material above a certain stage caused the gas to exhaust to the free face so that Te became dependent on the burden and not the stemming length.

Laboratory studies have indicated that the ideal stemming material for a 25-100mm diameter borehole is crushed rock with dimensions one tenth of the hole diameter. (Anon. 1988). The limestone used to prevent dust explosions should be around 100 micrometres (0.1mm) in size and is not

suitable as stemming material. The paradox arises that comparatively coarse stemming, which in an ineffective inert material, may reduce the exhaust gas temperature and prevent dust ignition while the ideal inert finely crushed limestone may lead to high exhaust gas temperatures and a dust explosion.

Several advantages arise when the correct type and mass of stemming material is used in a well designed blast. These include good fragmentation, minimum airblast over-pressure, reduced back-shatter and a low explosive consumption.

The use of detonating cord in blast holes and as connecting lines between holes may lead to an increased risk of a dust explosion. Exhaust gases may vent through channels in the stemming column caused by the detonating cord and the cord on the surface may disturb previously deposited dust. Most manufacturers now produce a low energy downline system which will minimise stemming disturbance.

The temperature and volume of the explosive gases may be reduced when a blast hole is stemmed with water or another heat absorbing substance. With water stemming a flat self-sealing plastic tube is inserted into a blast hole and water used to inflate the tube to the diameter of the blast hole. A commercial inert product with a gel-like consistency and packaged in plastic tubes is also available as stemming material. The composition of the product is designed for maximum heat absorbing properties.

7.9 REDUCED OXYGEN CONCENTRATION

A reduction in oxygen concentration causes an increase in dust ignition temperature and a decrease in the explosion parameters Pmax and (dP/dt)max. A minimum oxygen

concentration exists below which a dust explosion cannot develop regardless of the dust concentration or the ignition energy. The minimum oxygen concentration for a coal dust explosion is around 15 percent and it is likely that pyrite will require at least this oxygen concentration for a dust explosion to develop.

Two methods are available to reduce the oxygen concentration in an underground opening:

(i) Displace air with the inert explosion gases produced during detonation of high explosives;
(ii) Introduce an inert gas into the mine opening before blasting.

One kilogram of ANFO produces around 1.2m^3 of inert gas at normal temperature and pressure and similar volumes of inert gases are produced by other commercial explosives. The reduced oxygen concentration in an excavation depends upon the mass of explosive detonated and the excavation geometry. In the model of a stope blast considered in Section 6.4 the oxygen concentration was reduced to 19 percent when one hole was fired and 12 percent when three rows of six holes were fired. In the initial stages of a stope the void volume is comparatively small and one or two charged holes may provide sufficient inert gas to reduce the oxygen content below the minimum concentration necessary for an explosion. As the stope is mined out the void space increases and a large mass of explosive will be required to produce a sufficient volume of inert gases to prevent an explosion.

In a development drive the void space can be considered constant and the cut holes discharge dust and hot gases into a normal atmosphere. As successive holes are fired the inert gases displace air from the end of the drive and the oxygen concentration falls. The oxygen content in the first

20m of the 3m by 3m drive discussed in Section 6.5 fell to 16 percent after six holes loaded with a total of 30 kg ANFO were fired.

The use of an inert gas or foam to displace air from the end of a drive would inhibit the development of a dust explosion when the cut and easer holes are fired. In the 3m x 3m drive the introduction of 30 cubic metres of nitrogen would lower the oxygen concentration to 15 percent in a 13m length of drive. Alternatively a stable foam generated with an inert gas could be used to displace air from the face and the cut holes would be fired into an inert and heat absorbing atmosphere.

7.10 FILTER AND OXYGEN SELF-RESCUERS

FINAL LINE OF DEFENCE

Despite all the precautions taken in coal mines to prevent gas and dust explosions these events continue to occur frequently. The final line of defence in a coal mine is to provide all underground personnel with carbon monoxide filter self-rescuers. The unit is designed to allow miners to proceed through a respirable atmosphere containing high levels of carbon monoxide to a fresh air base or refuge bay.

In recent years oxygen closed-circuit self-rescuers have been designed for use in underground mines. The unit is worn on a belt and when activated supplies the wearer with an independent supply of oxygen for 30 to 60 minutes duration depending on the breathing rate.

Sulphide dust explosion in hard rock mines are usually associated with the use of high explosives and no hazards exist when a mine is evacuated before a blast. A danger

does exist that miners may return to sections of the mine containing unexpected sulphur dioxide concentrations. An even greater danger of gas poisoning exists when development and secondary blasting are permitted during a shift. The issue of filter or oxygen self-rescuers would reduce the risk of toxic gas inhalation.

FILTER SELF-RESCUERS

It has been reported that of all deaths attributed to colliery explosions or fires eighty to ninety percent are directly caused by carbon monoxide poisoning. In most coal mining countries legislation requires all underground personnel to carry with them a carbon monoxide self-rescuer.

The self-rescuer has been specifically designed to protect the wearer from carbon monoxide concentrations up to 1.5 percent in an otherwise respirable atmosphere for a duration of at least one hour. In recent years the self-rescuer has been responsible for saving an untold number of lives following major gas and dust explosions.

The function of a filter or canister respirator is to remove from the inhaled air any entrained respiratory poisons. They are designed to retain respiratory poisons by a screening action and self absorption, or to render them chemically harmless. A basic requirement of filter and canister units is that the cleaned air is suitable for breathing, i.e. it must contain at least 16 percent oxygen. Moreover, the air to be cleaned must not contain a higher proportion of poison than can be handled by the unit. The limit of usefulness of canisters against gas is taken as an average of two percent by volume although this figure may vary with the type of filter or gas. The material contained in a canister can only absorb or render harmless a certain

mass of gas and as a consequence the units become exhausted after a time depending on the gas concentration and breathing rate. In some units the reactive material may be poisoned by water vapour and the effective life reduced.

The carbon monoxide filter self-rescuer shown in Figure 7.5 is typical of the units available for use in coal mines. The self-rescuer is housed in a stainless steel case, weighs around one kilogram and is worn on a miner's belt. The catalyst used in the unit, called hopcalite, is a mixture of copper and manganese oxides and converts carbon monoxide to carbon dioxide at ambient temperature. The unit is designed to protect the wearer against carbon monoxide concentrations up to 1.5 percent in an atmosphere at 24°C and fully saturated with water vapour.

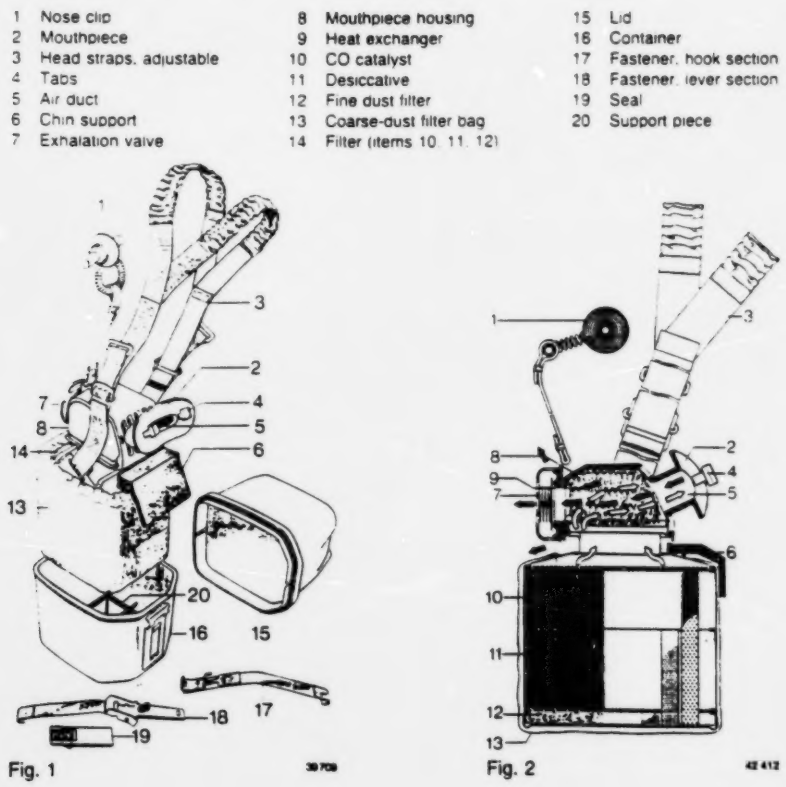


FIGURE 7.5 CARBON MONOXIDE FILTER SELF-RESCUER

Commercial filters are available for removing sulphur dioxide and hydrogen sulphide from the inspired air by absorption or catalytic conversion on specially treated activated carbon filters. One available filter can absorb 5000ppm of sulphur dioxide for 100 minutes at a flow rate of 16 litres per minute. The filter weighs less than 0.3 kg and is designed for use with a full face mask.

SELF-CONTAINED SELF-RESCUER

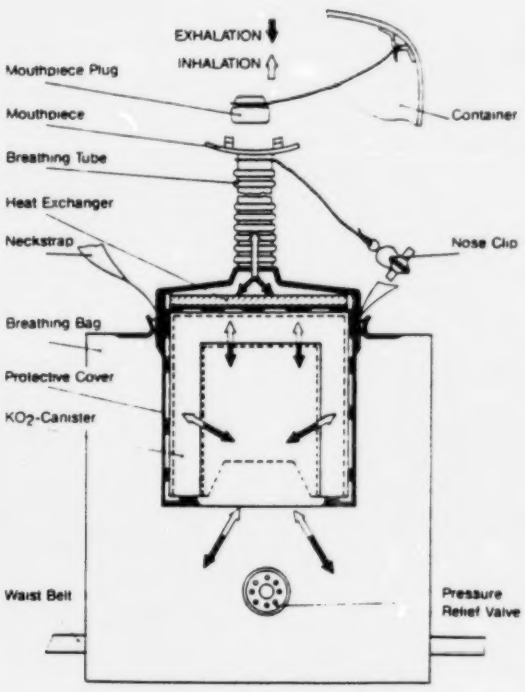
The limitations of the open-circuit filter rescue units have been overcome in units of closed-circuit design. Two types of closed-circuit units are available. The compressed oxygen type removes carbon dioxide from the exhaled air in a regenerative canister and replace the oxygen consumed during respiration from a compressed oxygen supply. The chemical oxygen type generates oxygen from the reaction of potassium superoxide with carbon dioxide and water vapour in the exhaled air. The simplified reaction formulae are as follows:

$$2KO_2 + H_2O = 2KOH + {}^3/_2\ O_2 + \text{heat}$$
$$2KOH + CO_2 = 2K_2CO_3 + H_2O + \text{heat}$$
$$2KO_2 + CO_2 = K_2CO_3 + {}^3/_2\ O_2 + \text{heat}$$

As potassium superoxide is the oxygen source and the carbon dioxide absorber at the same time, the resultant apparatus is of simple design, requires no moving parts and consists of few main components.

Chemical oxygen units have been approved for use underground in several countries and the self-contained units are designed to be worn on the miner's belt. Figure 7.6 is a schematic diagram of a self-contained self-rescuer. The unit, housed in a stainless steel container, weighs around 2 kg and measures 180mm high, 180mm wide and 100mm

deep. The unit weighs around 1.3 kg when removed from the container and the generation of oxygen begins with the first exhalation into the breathing bag. Pure oxygen is released filling the breathing bag with oxygen-enriched breathing air for re-inhalation. The chemical reaction of the potassium superoxide produces more oxygen than is consumed and a relief valve allows surplus oxygen flow off but prevents



Schematic Design and Air Flow

4.1 Self Rescuer in Container

- Weight	approx. 2 kg
- Dimensions:	Height approx. 180 mm Width approx. 180 mm Depth approx. 100 mm
- Carrying mode:	on belt
- Storage conditions:	preferably normal climate Temperature shall not be below 0 °C and above + 50 °C
- Service conditions:	Temperature of the unit at start above 0°C and below + 50 °C Ambient temperature during use above - 20 °C to + 50 °C
- Life expectancy:	a) **3 years** for apparatus carried along continuously (on the person, on machines or vehicles) and for apparatus kept in storage at elevated temperatures (max. 50 °C) b) **5 years** for apparatus kept in storage without mechanical stress at a storage temperature of max. 30 °C. c) maximum storage life **10 years** - for apparatus according to para. a: if after 3 years a random sample check is conducted annually - for apparatus according to para. b:

FIGURE 7.6 SCHEMATIC DESIGN AND AIR FLOW OF OXYGEN SELF-CONTAINED SELF-RESCUER

ambient air from entering the circuit. The performance data for the MSA SSR30/100 has been stated as:

Weight without container	1.3 kg
Life when worn regularly	3 years
Temperature of inhaled air at medium work	30-60°C
Use at rest (10 l/m)	100 mins approx.
Use at medium work (30 l/s)	30 mins approx.
Use at heavy work (50 l/m)	15 mins approx.

Potassium superoxide is a highly reactive substance and two or three breaths are sufficient to initiate the reactions and enable the full protective effect of the self-rescuer to come into play. Drager incorporates an additional starter cartridge which supplies oxygen immediately and independently of respiration once the self- rescuer is in position. This initial oxygen supply would be useful when an atmosphere contains sulphur dioxide at a concentration which impedes inhalation.

8. POST-EXPLOSION EVALUATION

8.1 OBJECTIVES

Considerable information on the development of a major dust explosion can be gained by an evaluation of the conditions existing in a mine after an explosion. The affected areas should be examined as soon as possible and before any attempts are made to clean up or commence mining. A review of the conditions prior to the explosion and the information collected after the explosion may indicate the cause and magnitude of the explosion, paths taken by the explosion gases, and weaknesses in the ventilation system. The effectiveness or otherwise of techniques used in a mine to inhibit a dust explosion and restrict the flow of sulphur dioxide can also be evaluated.

Evidence should be collected so as to provide information on the magnitude and path of the air blast, temperature and path of the flame front, path of the explosion gases and the mass of sulphur dioxide generated.

8.2 MAGNITUDE AND PATH OF THE AIR BLAST

All evidence of static and velocity gas pressures such as displacement of equipment and damage to ventilation doors, ventilation ducting, air and water pipes etc. should be evaluated and recorded on mine plans. The velocity pressure of the air blast can be estimated from physical damage to installations and the distances various pieces of equipment have been moved

8.3 TEMPERATURE AND PATH OF FLAME FRONT

Burnt ventilation ducting, melted plastic insulation on blasting and other cables, burnt timber, scorched paint, etc. provides evidence of the path of the flame front. The

melting or ignition temperature of various items will indicate the minimum temperature of the flame front as it travelled in the mine.

8.4 PATH OF THE COMBUSTION PRODUCTS

The main products of combustion of a sulphide dust explosion are hematite, magnetite and sulphur dioxide. The hot mixture of gases and dust particles will travei some distance past the point where the flame front dies out and eventually the gases will exhaust under their own pressure through the mine orifices or will be picked up by the ventilation air. The gases leave in their path fine particles of hematite or magnetite. The presence of hematite is easily detected due to its red colour and magnetite, a black dust, can be detected with a small magnet. A simple procedure is to brush a small amount of dust onto a piece of paper and run a magnet under the paper to observe if any of the particles are magnetic. This procedure can be continued along a drive, at say 100m sample intervals, until no magnetic material is observed. At each sample location a small mass of dust should be collected for microscopic examination and chemical analysis at a later date.

The oxidation of pyrite to magnetite in a dust explosion results in the production of hollow spheres, called cenospheres, which are easily recognised under a microscope. The presence of cenospheres in samples collected after a sulphide dust explosion was first recognised by Enright in 1981. Spheres observed with a scanning electron microscope were analysed with an electron probe and found to contain a high percentage of iron with minor amounts of lead and zinc and little sulphur. Chemical analyses and physical tests confirmed that the spheres were magnetite (Enright, 1984). Dust samples collected after explosions in Canada, Sweden and Australia have all shown the presence of cenospheres

even when the dust was predominantly hematite. Typical cenospheres found after a dust explosion are shown in Photos 8.1 to 8.3.

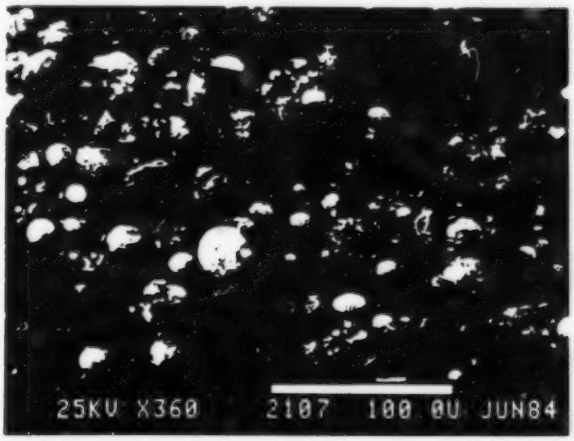


PHOTO 8.1 CENOSPHERES OBSERVED IN EXPLODED PYRITIC DUST

PHOTO 8.2 CENTRAL FIELD OF VIEW OF PHOTO 8.1 ENLARGED

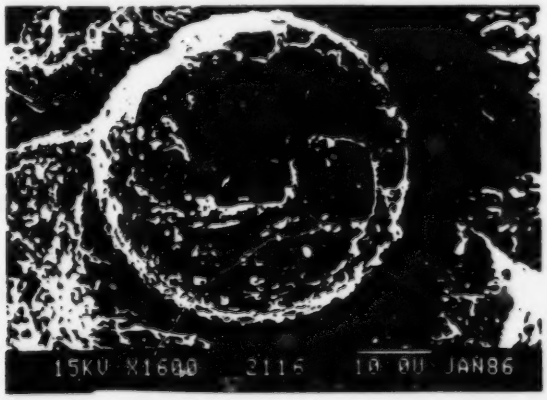


PHOTO 8.3 SEM PHOTO SHOWING HOLLOW CENOSPHERE

The presence of hematite or magnetite is easily detected near the origin of an explosion. As the turbulence and velocity of the explosion gases decrease the number of airborne dust particles decreases and the detection of a small number of settled hematite particles becomes difficult Magnetite produced during a dust explosion, either as a major component or mixed with hematite, can readily be detected even when only a small number of particles are deposited on existing mine dust. The magnetic particles in a dust sample can be separated readily and examined under a microscope to determine if they are cenospheres formed in a dust explosion or pyrrhotite found in normal mine dust.

8.5 VOLUME OF SULPHUR DIOXIDE PRODUCED

Cenospheres enable the path travelled by sulphur dioxide to be determined but provide no information on gas concentration. The gas concentration and air flow rate must be measured during and some time after an explosion in order to determine the total mass or volume of sulphur dioxide generated by the explosion. The actual mass of dust involved in the explosion can then be calculated using the mass of sulphur dioxide produced.

The installation of recording sulphur dioxide detectors in the return airways will provide data on the percentage of sulphur dioxide in the total air flow exhausting from a mine. Provided no air flow reversals occur in the intake and return airways during an explosion the total volume of sulphur dioxide can be estimated using existing ventilation surveys. The installation of recording static and velocity pressure instruments in the exhaust airways would permit a more realistic analysis of the changes in air flow and sulphur dioxide concentrations during and after an explosion.

The occurrence of a dust explosion underground would be rapidly detected by sulphur dioxide monitors in the exhaust airways. These instruments would provide warning of a dust explosion even when no other evidence was available to supervisors on the surface. The absence of sulphur dioxide in the return airways would indicate that a major dust explosion had not occurred following an underground blast and a mine could be declared free of sulphur dioxide in a comparatively short time. The installation of remote sulphur dioxide detections in strategic locations near the stope or development blast would ensure that no one entered an area where a small dust explosion, not detected by the monitors in the return airways, had occurred.

Valuable information can be gained after a dust explosion even when sulphur dioxide detectors are not installed in a mine. Readings with gas detecting tubes can be made in the exhaust airways after an explosion and the mine's rescue team can record details of sulphur dioxide concentrations as the mine is examined. Values recorded at regular intervals at major ventilation junctions would provide some idea of the volume of sulphur dioxide in the mine. Surface observations of air flow reversals and objects discharged from airways could be used to estimate extraordinary air flows.

9. RESEARCH INTO SULPHIDE DUST EXPLOSIONS

9.1 INTRODUCTION

Pure and applied research into coal dust explosion phenomena has been carried out at Government and industry supported Institutes since the early twenties. Large explosion galleries ranging up to 9 feet in diameter and 1200 feet in length have been constructed and full-scale experimental mines developed in Germany, France, Poland, Russia, Japan and the United States.

Little research has been devoted to sulphide dust explosions and a concentrated effort must be made if the problems unique to these explosions are to be solved. Fortunately the hard rock mining industry has one major advantage over the coal industry. Underground excavations can be used for research purposes and dust explosions created in drives with minimum damage to existing installations and no risk of a secondary dust explosion. Such an occurrence in a working coal mine would be impossible. Thus, in effect, every producing mine is a potential experimental mine where investigations can be carried out at minimum cost using existing excavations and machinery. There is a wide variety of expertise and equipment available at a mine which could be used to investigate dust explosions without causing undue interference to the normal operations. The Mining, Ventilation, Geological, Assaying and Computing branches of the mine staff can provide invaluable assistance in investigating the problem of dust explosions.

No one simple method will be found to prevent dust explosions as the development of a particular explosion depends on the shape and size of the underground excavations

and the blasting techniques used in it. Methods used successfully to prevent a dust explosion in a drive or raise may be of little practical value in a large open stope. Different techniques may be required during the life of a stope as the void space changes with rock extraction.

If the history of explosion prevention techniques developed by the coal industry can be used as a guide to the best approach in solving similar problems in the hard rock industry, then the answer lies in applied research with suppression techniques evaluated under practical mining conditions. Management must be prepared to actively support and encourage "in house" research and not just merely tolerate such activities on the mining lease.

To develop techniques to prevent dust explosions in the minimum time a concentrated effort must be made by the mining companies, Government and private Research Institutes, Universities and Tertiary Colleges. Close liaison must be maintained between the various organizations to provide mutual support, prevent duplication of research efforts and sustain motivation.

9.2 AREAS OF INVESTIGATION

A dust explosion requires fuel, an ignition source and oxygen and such an explosion can be prevented by any one of the following techniques:

* Remove or inert the dust or prevent the formation of a dust cloud.
* Prevent ignition of the dust.
* Reduce the oxygen concentration so as to prevent combustion of the dust.

The current knowledge of sulphide dust explosions is such that no one preventative technique appears to be superior or show more promise than another. Research must be directed to all of the above techniques and not concentrated on one specific area at the expense of the others.

Sulphide dust explosions can be broadly classified as one of two main types, explosions in development excavations and explosions in large stopes. Research should be directed to gaining a basic knowledge of dust explosion phenomena and applied research to explosions in development excavations and stopes.

The research programmes suggested in the following sections have been broadly classified as being suitable for laboratories, operating mines and experimental mines.

9.3 LABORATORY SCALE INVESTIGATIONS

(a) Explosion parameters of sulphide minerals.

- (i) Pmax and (dP/dt)max and minimum ignition temperature of:
  - Pyrite
  - Pyrrhotite
  - Chalcopyrite
  - Sphalerite
  - Galena
- (ii) Pmax and (dP/dt)max for mixtures of above materials

(b) Moisture content

- (i) Pmax and (dP/dt)max at various moisture percentages
- (ii) Generation of hydrogen sulphide with wet dust

(c) Reduced oxygen concentration

- (i) Pmax and (dP/dt)max at reduced oxygen concentrations

(ii) Minimum explosive concentration at reduced oxygen concentration

(iii) Minimum ignition temperature at reduced oxygen concentration.

(d) Frictional ignition of pyritic dusts.

(e) Comparison of sulphur analysis and mineralogy of ore with mine dust.

(f) Temperature and volume of exhaust gases generated by various commercial explosives.

(g) Efficiency and life of "sulphur masks" issued at some mines.

9.4 RESEARCH AT OPERATING MINES

(a) Mass of dust.

(i) Mass of settled dust on the walls, roof and floor of an open stope.

(ii) Mass of dust produced during and after a stope blast.

(iii) Mass of dust produced when the cut, easer and remaining holes of a development heading are fired.

(b) Atmospheric analysis

(i) Analysis of a stope atmosphere during and after a stope blast.

(ii) Analysis of the atmosphere near the face of a development heading as the cut, easer and other holes are fired.

(c) Evaluation of limestone dispersion techniques suitable for use in development headings.

(d) Investigation into the behaviour of water and inert stemming in development blasts fired in an open-cut using cine photography.

(e) Evaluation of various types of water sprays and foggers.

9.5 RESEARCH AT AN EXPERIMENTAL MINE

An experimental mine can be a fully instrumented specially designed mine currently in use or a section of an operating mine where less sophisticated tests can be carried out.

(a) Instrumented Experimental Mine

For a development blast determine the static and velocity pressure of air following the shock wave, the flame temperature and velocity, dust and gas analyses of the atmosphere.

(b) Section of an operating mine

In a section of an orebody where sulphide dust explosions have occurred frequently the objective would be to induce similar explosions on a regular basis for experimental purposes.

(i) Determine the mass of sulphur dioxide generated by a dust explosion and calculate the mass of dust involved in the explosion.

(ii) Determine the temperature of the combustion products of an explosion.

(iii) Systematic tests to determine the value of:

* limestone
* water and inert stemming
* conventional stemming
* water foams
* inert foams
* water sprays.

9.6 RECORDS AND POST-EXPLOSION INVESTIGATIONS

When blasting occurs in a reactive orebody it is as important to know the circumstance when a dust explosion does not occur as when an explosion does develop. Accurate records may indicate the conditions when dust explosions are more likely to develop and the measures most effective in preventing these explosions. The records should be simple, accessible and contain information on the ore blasted, blast design, mass of explosives, stemming, delays, void volume, drawpoint conditions, and preventive measures in force.

A post-explosion evaluation along the lines suggested in Section 7 should be carried out after each serious dust explosion. All available data on minor explosions should be recorded as soon as possible after the explosion. The information collected should include details of the flame front travel, air blast path, path of the explosion gases as indicated by hematite or magnetite, gas analysis, and production time lost to the explosion.

The installation of sulphur dioxide recorders in the exhaust airways would provide valuable information on all dust explosions and a means of detecting an explosion where no other evidence is available to personnel on the surface. This early warning may prevent miners from entering a potentially toxic atmosphere in the mine.

10. DETERMINATION OF DUST EXPLOSION PARAMETERS

10.1 INTRODUCTION

At the present time it is not possible to theoretically predict the explosion hazards of dust. As a consequence a range of explosibility tests have been developed in several countries. Many of the tests are similar in nature but a lack of standardisation has led to significant variations in the test procedures and consequently the results.

None of the explosions or ignition parameters such as minimum ignition temperature, minimum ignition energy, minimum explosive concentration, maximum explosion pressure and maximum rate of pressure rise are fundamental properties of matter. These parameters vary with the design of the experimental equipment, test procedures and the criteria established by the experimenter. For example, the minimum explosive concentration depends on the design of the explosion chamber, the degree of turbulence and the energy of the ignition source. The explosion parameters are also dependent on the particle size distribution, particle shape, surface area and degree of oxidation of the dust tested.

The following parameters of a dust sample are usually determined in laboratory scale tests.

- minimum ignition temperature
- minimum ignition energy
- minimum explosive concentration
- maximum explosion pressure and maximum rate of pressure rise
- minimum oxygen concentration for ignition.

10.2 MINIMUM IGNITION TEMPERATURE

The minimum ignition temperature of a dust suspension is the lowest temperature at which it will ignite spontaneously and propagate flame. The test is usually carried out in the Godbert-Greenwald furnace which incorporates an electrically heated quartz tube 216mm in length and 37mm in diameter (Dorsett et al. 1960). A weighed sample of dust is placed in a horizontal dispersion tube and blown by a volume of compressed air into the furnace (Figure 10.1). The temperature is measured by a thermocouple at mid-height in the furnace 0.8mm from the tube wall. The dispersing air is stored in a small reservoir and pressure in the reservoir can be varied to alter the dispersal conditions.

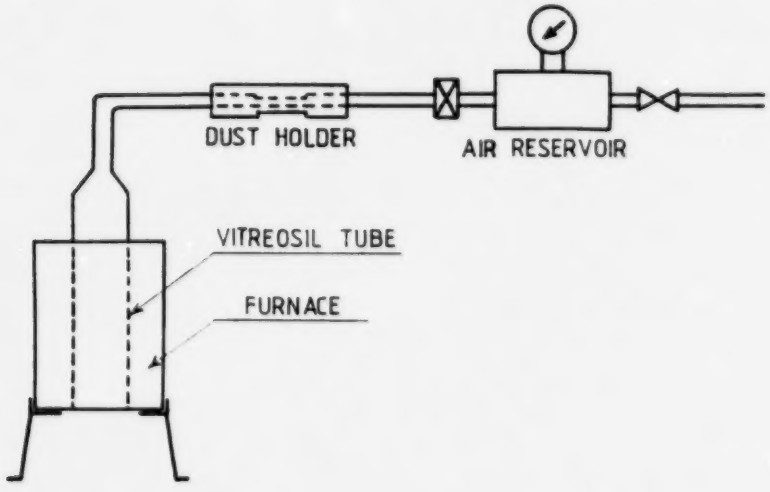


FIGURE 10.1 GODBERT-GREENWALD FURNACE

Ignition is indicated by the appearance of flame below the mouth of the furnace. The ignition temperature is the minimum furnace temperature at which flame appears at the bottom of the furnace in one or more trials in a group of four. The mass of dust and dispersal conditions are then

varied until the minimum ignition temperature is defined. The increment of temperature variation is 5°C and the highest temperature attained in the furnace is around 1000°C.

The Godbert-Greenwald furnace is used to determine the minimum ignition temperature in the United States, Norway and United Kingdom.

10.3 MINIMUM IGNITION ENERGY

The basic principle of most techniques used to study minimum ignition energy is to disperse the dust into a cloud which then passes between 2 electrodes. A spark of known energy is passed between the electrodes and the minimum energy required to ignite the dust cloud is observed. The minimum ignition energy of a dispersed cloud in the United States and the United Kingdom is determined in a Hartmann tube (Dorsett et al. 1960). This apparatus, Figure 10.2, consists of a "perspex" combustion tube 305mm long, 64mm inside diameter. Dust is placed in a hemispherical cup and dispersed by air passing through a tube on the axis of the cup, which in turn is dispersed by a mushroom-shaped deflector. The dispersing air is released from a reservoir of fixed volume, the pressure of which can be varied.

In the original Hartmann tube an electrical capacitor charged at low voltage (100 to 500 volts) is discharged through a step-up transformer to produce of 10,000 to 50,000 volts across the air gap of the electrodes. Ignition is denoted by flame propagating 100mm or longer in the tube. The energy of the spark is reduced until no explosions occur in at least 10 consecutive tests. Single sparks having theoretical energies of between 3 mJ and 2J are obtained by

changing condensers of capacity C with a constant voltage V, and then discharging them through the transformer. The minimum ignition energy E, is calculated from the expression $E = 0.5\ CV^2$.

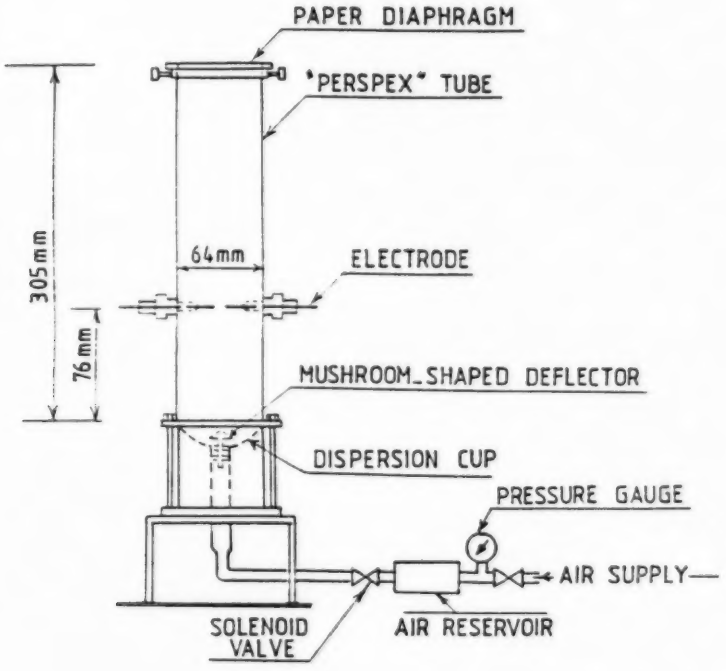


FIGURE 10.2 HARTMANN TUBE

There are two major criticisms of the use of this equipment for minimum ignition energy. The more important criticism is that it is assumed that the energy in the spark is the same as the total energy stored in the capacitor. This assumes that all the energy in the capacitor is dissipated and ignores energy loss in the transformer. The second criticism is that it is not possible to control the timing of the sparks accurately with respect to dispersal of the dust cloud. Several measurements must be made to allow for variations in the cloud concentration with time. An alternative method of determining the minimum ignition energy has been published in a new British Standard (BS 5858, 1980). The major difference from the method already described is in the circuitry used to produce the sparks.

The test currently used to determine the minimum ignition energy in Norway is derived from the findings of Eckhoff (1975). The test is based on the fact that if the discharge time of the sparks is long enough to reduce the shock "wave" effect on the dust in its vicinity, then very low ignition energies for dusts can be determined. Eckhoff's results indicate that the original Hartmann-tube system overestimated energies by a factor between 2 and 5. Unfortunately the only extensive lists of minimum ignition energies are those determined with the Hartmann apparatus and published by the United States Bureau of Mines and the British Fire Research Station.

10.4 MINIMUM EXPLOSIVE CONCENTRATION

In the United Kingdom the minimum explosive concentration of a dust is taken as the minimum mass of dust suspended in a given volume that is capable of igniting and sustaining flame propagation. This parameter is measured in the "perspex" Hartmann vertical tube apparatus previously described with an electric spark as the ignition source.

The top of the tube is fitted with a paper diaphragm held in place by a locking ring. A known mass of dust is dispersed around a continuously activated electric spark. Low dust concentrations may produce very small discrete flames which are difficult to observe and the actual value is therefore subject to variations due to the interpretations of different operators. Tests are carried out with reduced amounts of dust until no flame propagation is observed in ten consecutive tests for a given concentration.

The criterion for ignition in the United States is that the filter paper diaphragm should burst in at least one of four successive tests, this normally being associated with an explosion pressure of about 20 kPa. The appearance of a

flame in the tube without the bursting of the diaphragm is not considered as an explosion, unlike the criterion adopted in the United Kingdom.

In Norway the minimum explosive concentration is determined in a 15 litre explosion vessel using a strong ignition source (100-200 watts). In cases where the dust is difficult to ignite a more powerful ignition source may be used. The criterion for an explosion is that flame propagating at least 100mm must be observed. The dust concentration is determined by reproducing the dispersion conditions required for ignition and then extracting 200ml of the dust suspended in the vicinity of the ignition source. The concentration of dust which gives an explosion probability of 0.5 is regarded as the minimum explosive concentration (Eckhoff, 1978)

The minimum explosive concentration is determined in Switzerland using a 20 litre sphere or a $1m^3$ vessel. In the 20 litre test the minimum explosible concentration is determined by the smallest mass of dust to produce distinct trace on the pressure-time record or by study of the residue which indicates an explosion. Confirmation of this value is sometimes obtained in a test in the $1m^3$ vessel.

10.5 EXPLOSION PRESSURE AND RATES OF PRESSURE RISE

The maximum explosion pressure, rate of pressure rise and optimum explosive concentration for dust clouds are determined by dispersing and igniting a known dust concentration in an explosion chamber. The dust is usually dispersed by a jet of compressed air and the air introduced during the dust dispersion creates turbulence, increases the initial pressure and the amount of oxygen available for combustion. The greater the quantity and pressure of air introduced, the higher the maximum pressure and rates of

pressure rise produced by the explosion. The igniting source may be a single timed spark, a continuous electrical spark, a heated coil, or the flame produced by ignition of a chemical compound such as nitrocellulose. The pressure rise is measured by a pressure transducer.

A schematic drawing of the Hartmann apparatus which incorporates a 1.2 litre cylindrical explosion tube is shown in Figure 4.1. A non-return valve is fitted on the inlet to the explosion tube to prevent the escape of gases during the test. A known mass of dust, initially around 0.1g, is placed in the dust cup and the dust is dispersed by the release of 50 ml of air pressured to 700 kPa. In determining the concentration under test it is assumed that the dust is uniformly dispersed into the full volume of the tube. Tests are normally carried out at dust concentrations of 100, 250, 500, 750, 1000, 1250, 1500, 2000 and 3000 g/m^3. Pressure developed in the explosion chamber is usually measured by a piezoelectric pressure transducer coupled to a charge amplifier. The signal from the charge amplifier is then fed to a storage oscilloscope and a chart recorder.

A typical pressure-time record is given in Figure 10.3 in which the small initial pressure P_1 is due to the entry of the dispersing air, the pressure rise due to the explosion normally being more pronounced. Activation of the ignition source was at time t_0, dust dispersion at t_1 and the explosion commenced at t_2. The following values may be calculated from the pressure time record.

$$\text{Ignition delay} = t_2 - t_1$$

$$\text{Maximum explosion pressure} = p_2 - P_1$$

$$\text{Maximum rate of pressure rise} = \frac{\Delta P}{\Delta t}$$

$$\text{Average rate of pressure rise} = \frac{P_2 - P_1}{t_3 - t_2}$$

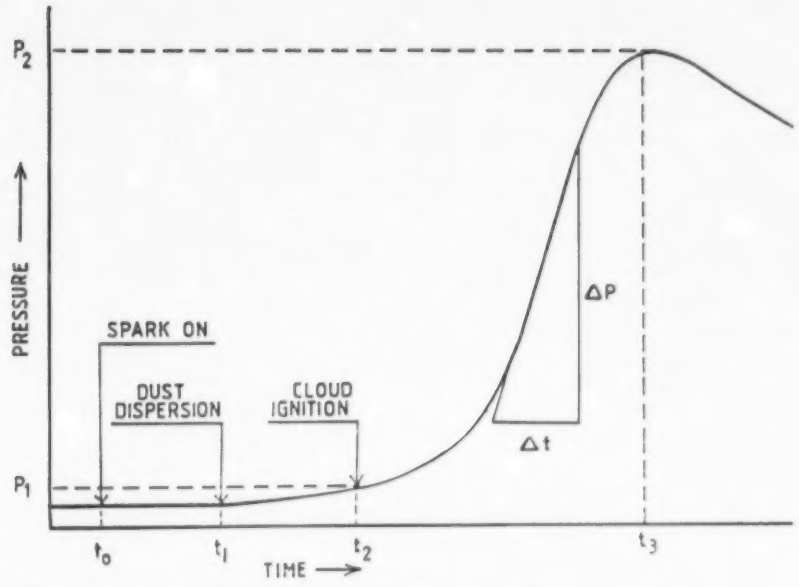


FIGURE 10.3 PRESSURE-TIME TRACE

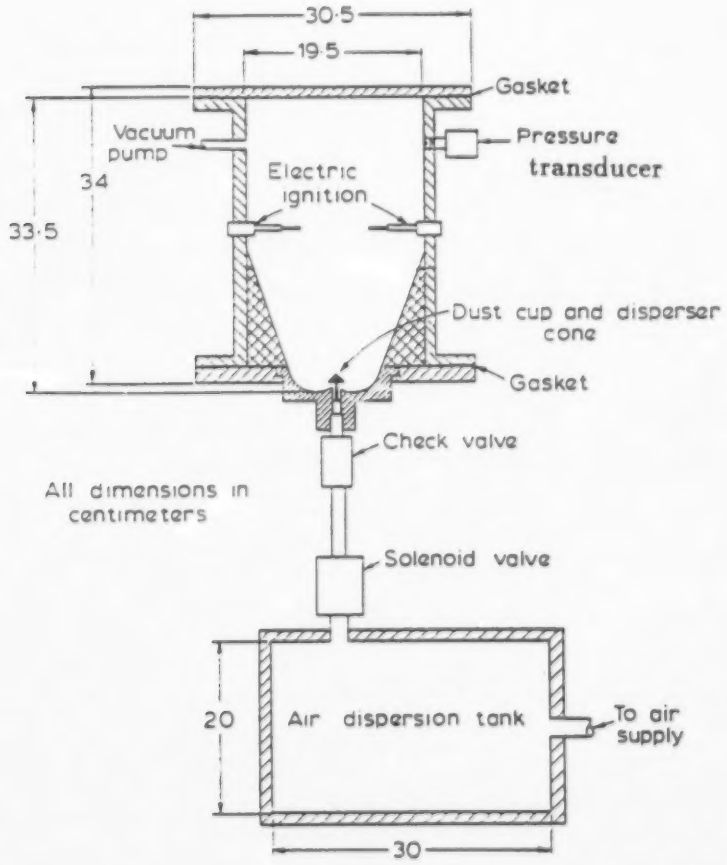


FIGURE 10.4 VERTICAL CROSS SECTION OF THE 8-LITRE EXPLOSION CHAMBER (Hertzberg et al. 1979)

Cybulski, (1975) considered that the average rate of pressure rise between 0.1 P_2 and 0.9 P_2 removed any interpretation in determining P_1. A number of tests are carried out at the same concentrations to ensure that the maximum values representing the worst conditions, are determined. The Hartmann apparatus is used in the United Kingdom and Norway to determine the maximum explosion pressure, Pmax, and maximum rate of pressure rise, (dP/dt)max. Other laboratory chambers used to determine Pmax and (dP/dt)max range from 0.008 to 1.0m^3 in volume and vary in shape with cylindrical, spherical and rectilinear vessels in use. The techniques used to disperse the dust also vary with changes in chamber volumes. An 8 litre explosion chamber with a dust dispersing technique similar to that used in the Hartmann chamber has been developed by the United States Bureau of Mines (Hertzberg et al. 1979). A section through the chamber and the air dispersion tank is shown in Figure 10.5.

In Switzerland Pmax and (dP/dt)max are determined in a 20 litre spherical chamber developed by Bartknecht (1981), although more detailed investigations may be made in a 1m^3 vessel. In the 20 litre sphere (Figure 10.5), a known amount of dust is placed in the storage chamber and is pressured in air to 2000 kPa. The sphere is evacuated so that atmospheric pressure is reached when the dust is dispersed from the storage chamber through the ring-shaped perforated nozzle. A 10,000 J chemical igniter is used in the centre of the chamber. A 60 ms ignition delay ensures that the storage chamber has been emptied of dust and that all tests are carried out at a constant level of turbulence. Only dusts having a mean particle size less than 63 microns can be tested directly in the sphere; constant temperature conditions are maintained by a water pocket surrounding the sphere.

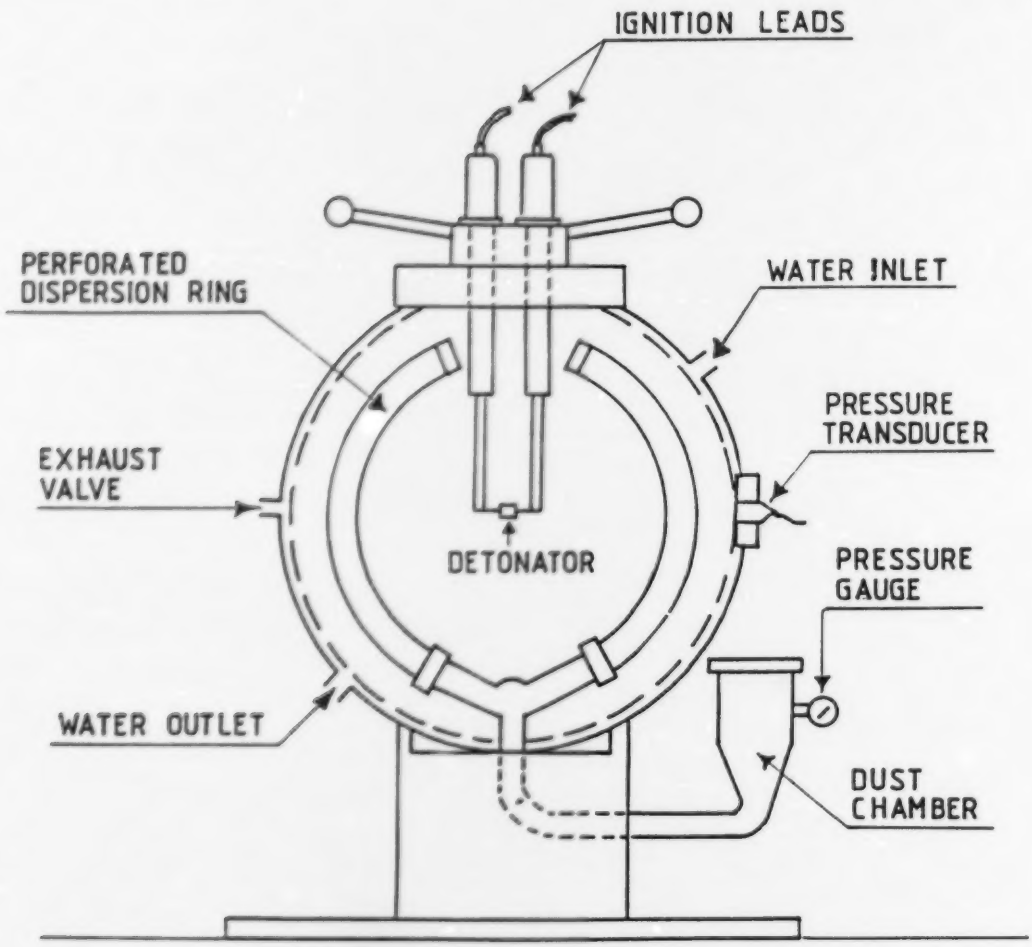


FIGURE 10.5 20 LITRE SPHERICAL EXPLOSION APPARATUS
(Bartknecht, 1981)

10.6 MINIMUM OXYGEN CONCENTRATION FOR IGNITION

In the United Kingdom and Norway the minimum oxygen concentration required to enable a dust suspension to explode is determined in a modified Hartmann tube apparatus with an inert-gas/oxygen mixture used to disperse the dust. The dust sample is placed in the dispersion cup and the top of the tube is fitted with a paper diaphragm to prevent the

ingress of air. The air in the tube is displaced by allowing the reduced oxygen mixture, usually oxygen and nitrogen, to percolate the system without disturbing the dust while the oxygen content in the tube is monitored. When the oxygen is reduced to the required level the ignition source is activated and the dust is dispersed into the tube by a blast of the reduced oxygen mixture. The criterion for ignition is the same as that used in determining the minimum explosive concentration. The oxygen is reduced until no flame propagation is observed in ten consecutive tests at constant oxygen concentration.

In the United States tests are carried out in controlled atmospheres of reduced oxygen to determine the level at which flame propagation is just prevented. The test is carried out either in a spark apparatus consisting of a glass tube 480mm in length and 38mm diameter with electrodes 400mm from the bottom of the tube or in a modified furnace. In the spark apparatus provision is made for the oxygen reduced atmosphere to flow through the spark tube continuously. The dust is dispersed into the tube by a blast of the controlled atmosphere and the ignition source is a continuous electric spark. The criterion for ignition is that flames should propagate at least 100mm from the spark (Dorsett et al. 1969).

The furnace test to determine the minimum oxygen concentration is carried out at 850°C in a modified version of the furnace apparatus used for determining the ignition temperature of a dust. Research by Tonkin (1972) indicated that this test gave values which were unrealistically low owing to the high operating temperature of the test.

10.7 APPLICABILITY OF LABORATORY SCALE TESTS TO FULL SCALE OPERATIONS

IGNITION TEMPERATURE

In the Godbert-Greenwald furnace the dust is dispersed by air at ambient temperature and consequently the temperature of the air within the furnace tube is reduced on entry of the dust. The mechanism by which the dust ignites will depend on the nature of the dust and is likely to involve ignition on the walls, or in the hot air. In addition the temperature inside the furnace may lead to the production of a gaseous mixture which in turn may be ignited by the hot air or furnace wall. The ignition temperature determined in this test would be reasonably accurate if ignition occurred by contact with the furnace wall, but is probably an overestimation if the somewhat cooler hot gases are responsible for igniting a dust suspension.

The minimum ignition temperature of a dust determined in small scale tests should generally be regarded as higher than that required to ignite the same dust in a large scale operation.

MINIMUM IGNITION ENERGY

For many years the standard apparatus used for this test was the Hartmann tube. In the past few years a number of design problems have been recognised in the apparatus and it is now considered that the test overestimates the minimum ignition energy.

A single minimum ignition energy test has not been accepted but for most practical purposes the objective of the test is to distinguish between dusts which are likely to be ignited by electrostatic or other low-energy sources and those which are not particularly sensitive. A knowledge of

the minimum ignition energy of coal or pyritic dust is of little interest when the ignition source is the flame front of a methane explosion or associated with high explosives used in mining operations.

MAXIMUM EXPLOSION PRESSURE

The maximum pressure developed by a gas or dust explosion in a closed vessel is independent of the vessel size and shape if heat lost to the vessel is disregarded (Nagy and Verakis, 1983). A significant heat loss will occur with resultant lower maximum pressures in ducts where the cross-sectional area is small relative to the length and in vessels having one or more walls near the ignition source. Bartknecht (1981) concluded that values of maximum explosion pressures determined in a 20 litre chamber were slightly lower than those measured in a $1m^3$ chamber.

MAXIMUM RATE OF PRESSURE RISE

The rate of pressure rise is affected by the vessel volume and shape, intensity of the ignition source, fuel-air ratio, and location of the ignition source. In non-spherical vessels with central ignition Ellis (1928) showed experimentally that the shape of the flame front tended to assume the shape of the vessel. Nagy (1983) derived the following expressions for an isothermal system assuming the flame front was identical with the shape of the vessel.

For spherical vessel $$\frac{dP}{dt} \propto \frac{1}{r}$$

For nonspherical vessels $$\frac{dP}{dt} \propto \frac{S}{V}$$

where S is surface area, V volume and r radius of the spherical vessel.

Bartknecht (1981) carried out a series of extensive tests in large vessels ranging from 1 to 60m³ in volume and deduced that provided conditions such as concentration, initial pressure, and ignition characteristics remained the same, the maximum rate of pressure rise was inversely proportional to the cube root of the vessel volume:

$$\text{i.e } \left(\frac{dP}{dt}\right)_{max} V^{1/3} = Kst$$

where $\left(\frac{dP}{dt}\right)_{max}$ is the maximum rate of pressure rise, V the volume of the vessel and Kst a characteristic of the dust.

Bartknecht (1981) has shown that the maximum rate of pressure rise can be considerably underestimated when determined in small vessels, such as the Hartmann apparatus, and that the direct scaling of this data to large volumes is generally unreliable. He concluded that the cubic law can only be applied when the minimum volume of the test vessel is 20 litres, a prescribed testing procedure is adhered to and sufficiently high ignition energies are used.

The reliability of small scale explosibility tests and their relevance to full scale tests has been investigated by the United States Bureau of Mines (Hertzberg et al. 1982). Various inert dusts were added to coal dust and the explosion parameters of the mixtures were measured by three experimental groups. Tests were carried out in an 8 litre chamber, a 2m diameter by 28m long explosion gallery open at one end and at the Bruceton Experimental Mine. The experimental data were found to be in good agreement with the two independent, full scale studies.

MINIMUM OXYGEN CONCENTRATION

Tests for the minimum permissible oxygen concentration are carried out either in the vertical tube apparatus or in the furnace apparatus at a test temperature of 850°C. Results reported using the furnace apparatus were found to be unrealistic at the Borehamwood Research Station (Field, 1982) as the minimum permissible oxygen concentration at which no ignition took place was significantly higher when the ignition source was a heated surface or furnace than when it was a spark. Craven (1967) found that an oxygen nitrogen mixture at 850°C could support combustion of terephthalic acid dust with a minimum oxygen content of 4 percent whereas an ambient mixture required a minimum of 12 percent.

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